G. Popov

THE WORKING OF MINERAL DEPOSITS

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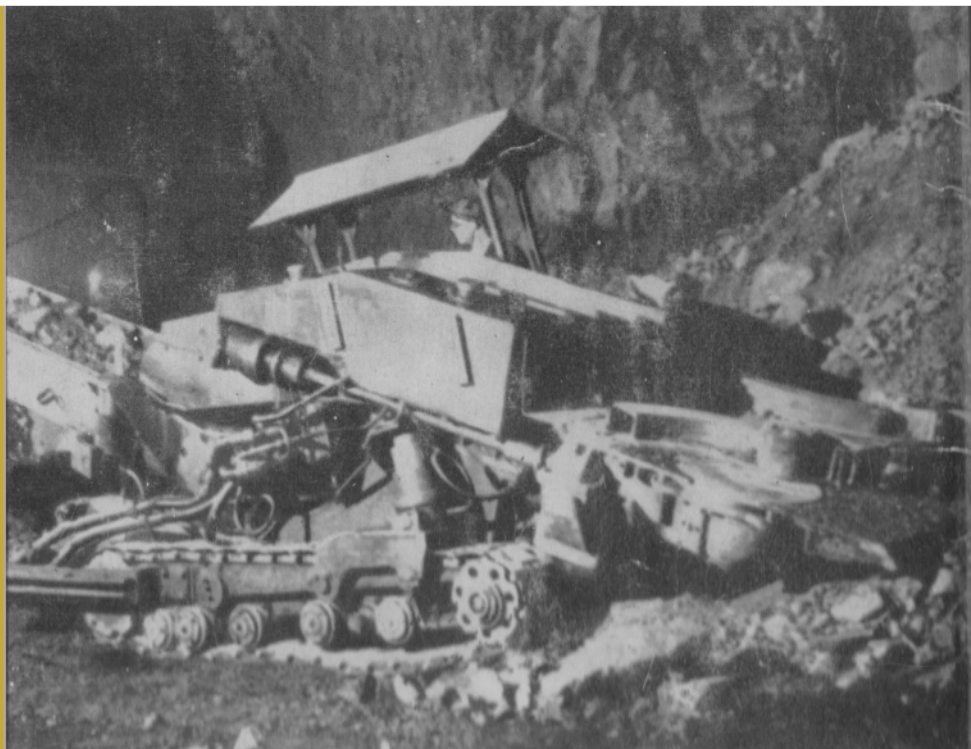

The text-book consists of two parts.

The first part deals with the underground working of mineral deposits. It covers the basic concepts and terms used in mining and treats on problems involved in the opening-up of sheet and ore deposits, the methods employed in the working of coal and placer occurrences. It also describes the correct ways of selecting most suitable methods for mining a given ore deposit depending on the mining, geological and technical factors, and offers a technical and economic comparison of the possible systems.

The second part is devoted to the opencast working of mineral deposits. It discusses the basic concepts, the mining terms, the choice of opencast methods of mining, the determination of the depth and production capacity of an open-pit, the major operations involved in open work, the early development of deposits, the technical and economic characteristics of opencasting, dredging and hydraulicking in placer deposits, new ways and main problems in the exploitation of mineral bodies.

The text is meant for students of higher mining colleges and mining departments. It may also be useful for engineering and technical staff in the field.

The book has seen two Russian editions and has been substantially enlarged by the author for the English edition.

Г. Н. ПОПОВ

РАЗРАБОТКА МЕСТОРОЖДЕНИЙ ПОЛЕЗНЫХ ИСКОПАЕМЫХ

ИЗДАТЕЛЬСТВО «НЕДРА» МОСКВА

G. POPOV

THE WORKING OF MINERAL DEPOSITS

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TO THE READER

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PART ONE

THE UNDERGROUND WORKING OF MINERAL DEPOSITS

I. THE DEVELOPMENT OF UNDERGROUND MINES

CHAPTER I

BASIC CONCEPTS AND TERMS

Study of the course "The Working of Mineral Deposits" requires mastering the relevant basic concepts and terms.

In this Chapter the reader will find the necessary definitions as adopted in the USSR.

1-1. MINES AND WORKABLE MINERAL DEPOSITS

A *mining enterprise* is an industrial unit carrying out the exploitation or exploration of mineral deposits.

A *mine* is an independent operational and economic unit of a mining enterprise.

A *mine-field* is a part of a mineral deposit allotted to one mine. Mine-fields differ in shape. In bedded coal deposits they usually have a rectangular shape, but with ore bodies their outline conforms to that of the body. One must distinguish the boundaries of a mine-field, viz. the upper boundary or up-dip limit, the lower boundary or down-dip limit, and the lateral boundaries or strike limits. A mine-field is not worked at once over its entire area but by separate sections—levels or panels—mined in a definite sequence (Figs. 1-3).

A *level* or *horizon* is that part of a mine-field limited along the rise and dip by level drifts, entries or roadways, and on strike by the boundaries (Fig. 1). Levels are usually mined in descending order. The interval between levels is said to have vertical (or true) height, or inclined height (the distance along the line of dip between the two drifts delimiting the level).

A *panel* is that portion of a mine-field marked out for extraction in working horizontal (bedded) and slightly inclined occurrences (see Figs. 2, 3).

With a gently dipping bed a panel is limited on the dip or rise by main (level) drifts or entries and by one of the boundaries of the mine-field, and along the strike by the boundaries of neighbouring panels. With gentle dip a panel is delimited by the contours of the

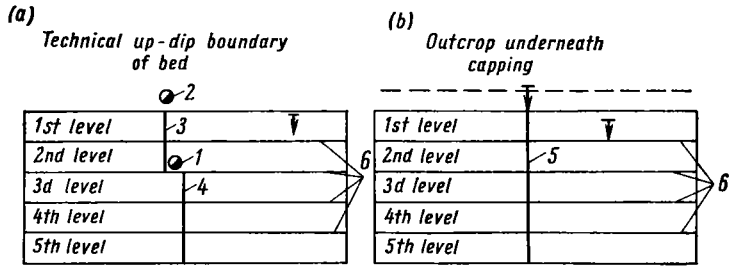


Fig. 1. The distribution of levels in a mine-field (in the plane of the bed):
(a) opened by a vertical shaft; (b) opened by an inclined shaft; *1*—main shaft; *2*—ventilating shaft; *3*—permanent gravity incline; *4*—permanent slope; *5*—inclined shaft; *6*—level entries, or drifts

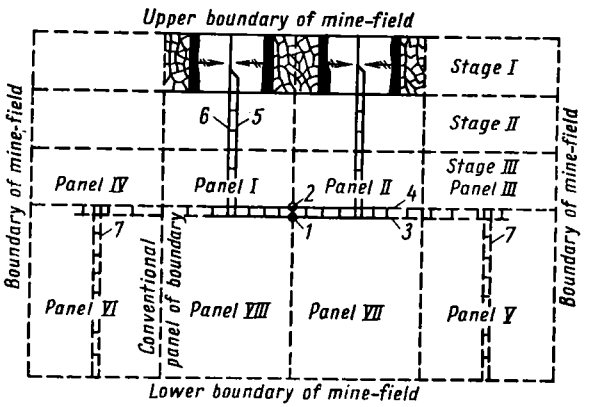


Fig. 2. The development of a mine-field with a gentle dip:
1—main shaft; *2*—ventilating shaft; *3*—main haulage road; *4*—main ventilating entry; *5*—panel brake incline; *6*—manway; *7*—panel slopes

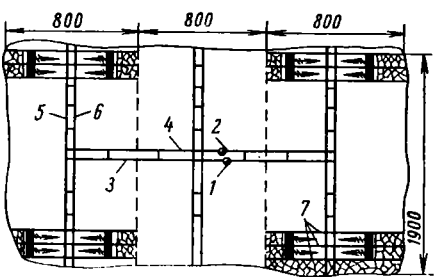


Fig. 3. Panel method of developing a mine-field in working a level seam:
1—main shaft; *2*—ventilating shaft; *3*—main haulage entry; *4*—main ventilating entry; *5*—haulage panel entry; *6*—ventilating panel entry; *7*—extraction or butt entry

mine-field and its boundaries with adjacent panels. Bedded deposits and some slightly dipping ones are mined by panels; steeply pitching and sloping deposits are mined by levels.

A *sub-level* is that part of a horizon (floor, level) delineated on the rise and dip by sub-drifts (sub-entries), and along the strike by the

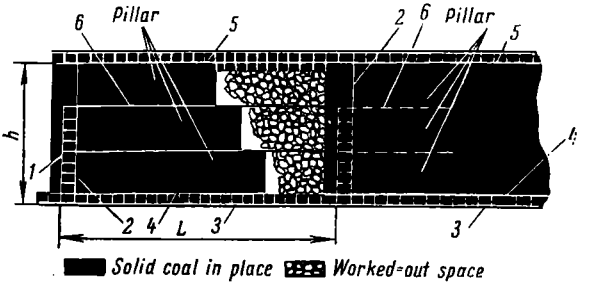


Fig. 4. Disposition of the winning area and producing sections (pillars) of a brake incline for long-wall mining of a gently dipping coal seam on strike (in the plane of the bed):

1—brake incline; *2*—manway; *3*—haulage entry; *4*—ventilation cross heading; *5*—ventilating entry; *6*—sub-entry; *h*—inclined level interval; *L*—length of the extraction area served by the brake incline

boundaries of the mine-field or by raises (for individual blocks). The number of sub-levels in one (level) ordinarily varies between two and four, depending upon the existing conditions and the accep-

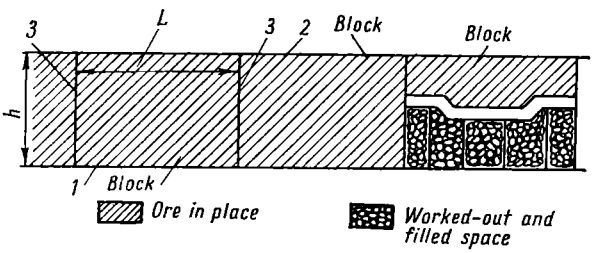


Fig. 5. Disposition of producing sections (blocks) in a thin, steeply dipping ore body mined by cut-and-fill methods:

1—haulage drift; *2*—airway; *3*—raise; *h*—inclined level interval; *L*—length of block

ted method of mining. With many systems of mining the level is excavated without being divided into sub-levels.

A *winning area* is that part of a level confined between two neighbouring gravity or brake inclines, permanent slopes, or dumping chutes. The winning area is designated according to the main working served, for example, gravity incline area, permanent slope or dumping chute area (Fig. 4).

A *producing section* is the portion of a winning area within which actual stoping is being carried on. In working bedded deposits producing sections are often called pillars (Fig. 4), in mining ore bodies they are usually called blocks (Fig. 5). Block extraction is ordinarily carried out without preliminary division of the level into winning areas.

1-2. TYPES OF MINING OPERATION, CLASSIFICATION OF MINE WORKINGS

Types of Mining Operation

The working of mineral deposits involves the following mining operations: capital work, development (or preparatory) work, subsidiary or subordinate work (also called development within the deposit, or first mining development), and finally stoping.

Capital work (*early development*, or *opening-up*) comprises the excavation of openings providing access from the ground surface to the deposit, or from some already exploited part of the occurrence to an over- or underlying unmined portion of it, and enabling the driving of levels or main drifts (depending on which development scheme is adopted, level or panel).

Development (or preparatory) *work* is that involved in driving mine workings within opened-up portions of a deposit to enable stoping operations to be started.

Subsidiary development (*blocking-out*) is work carried out in the developed portion of a deposit and having subsidiary significance for stoping operations.

Stoping is work done to extract minerals from the developed and blocked-out portion of a deposit. Two basic types of stoping are distinguished, the bulk or roundabout and resuing (or selective mining).

In bulk extraction different grades and classes of the useful mineral are extracted, without their separation.

Resuing is used in two instances: (1) when the useful mineral of one grade or class is temporarily left in the earth's crust; (2) when different grades and classes of the mineral are extracted simultaneously, but separately. The selective method of mineral extraction when barren rock enclosing the ore body is left in the stope area should also be regarded as resuing.

Classification of Mine Workings

Major, or *permanent* mine openings are driven to open up the entire or part of the mine-field. They include shafts, adits, shaft or bottom station workings (also called shaft inserts), special purpose

service rooms, crosscuts, lateral or fringe drifts, entries and roadways, permanent slopes, gravity or brake inclines, winzes, etc.

Development workings are excavated to prepare parts of the mine-field for stoping operations. They include auxiliary or district crosscuts, drifts (entries), crossdrifts (cross-entries), raises, winzes, gravity inclines, slopes and dumping chutes serving individual winning areas, along with prospecting or exploring openings driven to define more precisely the orientation of development workings.

Subordinate, or *temporary* workings are driven within production sections or units with the purpose of deploying stoping operations. These comprise: through-cuts, crossheadings, auxiliary drifts, slopes (dumping chutes), etc.

Stopes are productive workings where actual extraction of the mineral is done; they include production stopes and wall faces.

1-3. CLASSIFICATION OF MINERAL RESERVES

The Degree of Exploration

By reserves of solid minerals is understood their tonnage contained in the earth's crust within a definite area. Depending upon the extent of the reserves the areas containing them are classified as basins, regions, occurrences, mine-fields, etc. According to the degree to which they have been explored and studied the reserves of useful solid minerals are divided into three categories—A, B and C, the first being subdivided into A_1 and A_2, and the last into C_1 and C_2.

A detailed definition of these reserves is given in manuals on applied geology and prospecting. Elaboration of projects and appropriation of capital funds for the construction of mining enterprises are based on inventory resources of useful minerals (see below) belonging to categories $A_2 + B + C_1$, and in occurrences where exploration for resources of the A_2 category seems to be impracticable—on inventory reserves belonging to the categories $B + C_1$.

Reserves of the C_2 category are considered in long-term planning and in planning geological and prospecting work.

The correlation of the A_2, B and C_1 reserves necessary to substantiate projecting of construction work and the appropriate capital investment is stipulated by relevant instructions for the different mineral occurrences.

Subdivision of Mineral Reserves According to Industrial Use

Potential (geological) reserves. By geological reserves contained in the earth's crust are meant all discovered resources of a mineral occurrence within the boundaries of a clearly defined contour line

and depth irrespective of the percentage of its useful constituents and their possible industrial use in the immediate future.

Inventory reserves. These include such geological resources which by their quality and technical and economic standards meet the demands (standards and specifications) of the industry, their exploitation being commercially profitable at the present level of technical and economic development.

Other potential (geological) reserves, which cannot be referred to the inventory group, are termed extra-inventory resources.

Workable reserves. As stipulated by the instructions issued in the coal and metallurgical industries, workable or commercial reserves are such inventory reserves which are to be extracted and delivered to the ground surface according to the projects or plans for the exploitation of the particular mineral occurrence.

Workable reserves are determined by subtracting estimated overall losses from the total inventory resources.

Classification of Reserves as to the Degree of Their Preparedness for Extraction

Opened up, or actual reserves include that part of the mineral masses whose extraction has been made possible by the drivage of all early development workings envisaged in the pattern adopted

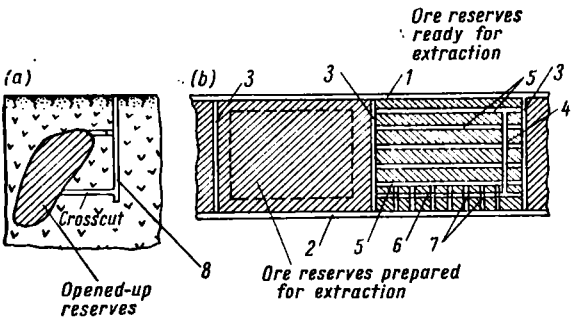


Fig. 6. Distribution of ore reserves according to their preparedness for mining: *a*—section (side view) of opened-up ore body; *b*—section of block on strike with ore body worked by open sublevel stope; *1*—vent drift; *2*—haulage drift; *3*—raise; *4*—cut-out raise; *5*—sublevel drift; *6*—grizzly drift; *7*—ore chutes; *8*—vertical shaft

for the opening-up of the deposit. Actual reserves ordinarily lie above the elevation mark at which the deposit is undercut by permanent mine workings and wherefrom development openings are subsequently driven (Fig. 6).

Mineral reserves in safety pillars under haulageways, surface structures and other edifices are considered separately and included

in the group of active reserves after the elimination of the protected units, or after the issuance by the Mines Inspectorate of a permit for the recovery of the safety pillar.

Prepared reserves comprise the minerals contained in winning areas and producing sections in which all development openings envisaged by the development scheme have been driven (Figs. 6 and 7).

Mineral reserves left in recoverable pillars (intervening, chain and arch ones) are not considered to be ready for extraction until

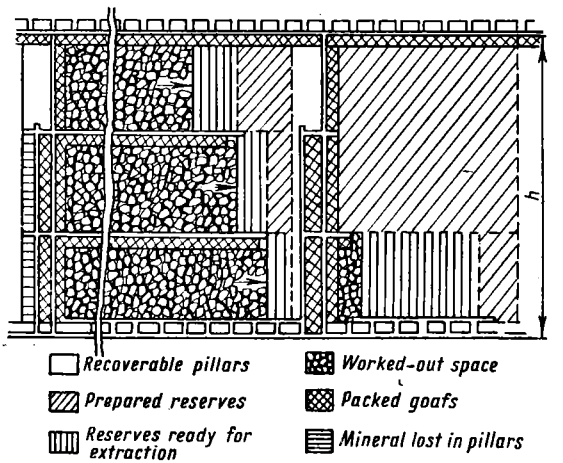


Fig. 7. Distribution of coal reserves according to the degree of their preparedness for working by breast-stoping with advance gravity inclines (section in the plane of bed):

h—level interval; arrows indicate direction of production face advance

after development openings have been made, mined-out spaces gobbed up and other operations performed, as envisaged by the plan for the recovery of the pillars in accordance with the adopted mining method.

Reserves ready for extraction include mineral masses contained in winning areas where subordinate development workings required for the initiation of stoping operations have been excavated (see Figs. 6 and 7).

When classifying reserves in the appropriate group according to the degree of their preparedness one should bear in mind that actual reserves are part of the inventory reserves, the prepared reserves part of the actual or opened-up reserves and those ready for extraction part of the prepared reserves. Building up of actual, prepared and ready-for-extraction reserves should be done in good

time, i.e. opening-up should precede development, the latter precede subsidiary development and this, in turn, precede stoping. Failing this, grave consequences may result, such as nonfulfilment of the production plan, disruption of the normal mine operation and rising production costs, and in some instances increased operational hazards. The necessary stocks should be built up at the very beginning of exploitation and kept up throughout the period of mining the deposit. To maintain permanent stand-by stocks of opened-up and developed reserves, simultaneously with stoping of this or that portion of the mine-field operations involving opening-up and development of other parts of the mine-field must be carried out well in advance of stoping operations.

Advanced opening and development work relative to stoping depends upon the nature of the mineral occurrence. In the case of geological disturbances (faults, folds, squeezes, bulges), demanding additional prospecting and corresponding time delays, the margin should be large. In estimating the amount of prepared carry-over reserves, usually expressed in terms of planned output over a period of several months, one should consider the sequence in mining the level (from the shaft towards the level boundaries, from the boundaries towards the shaft or synchronous working of the level along its entire length), the time needed for opening-up and development, or for development alone (with appropriately advanced opening), and the time required for stoping the level, due account being taken of the existing stand-by winning areas.

At all times the period of development or opening-up and development of a new level should not exceed that required for the stoping of the worked level, due allowance being made for the necessary lead in development work. Other types of reserves should be estimated in a similar way.

There are definite standards established in the industry for carry-over reserves. For instance, in mining ore bodies the accepted prepared carry-over reserves are within the range from 12 to 24 months' output of the mine, and those ready for extraction between 3 and 6 months' output. More detailed information on standard reserves is given in mining regulations concerning exploitation of coal and ore deposits and in special instructions.

CHAPTER II

OPENING-UP OF STRATIFIED DEPOSITS

2-1. CALCULATION TECHNIQUES IN SOLVING MINING PROBLEMS

By *calculation* (*analytical*) *methods* are meant all procedures for quantitative solution of mining problems.

The basic demands on the working of mineral occurrences are:

maximal economic efficiency and labour productivity should be ensured;

in mining a deposit the mineral wealth should be utilized to the fullest and with utmost expediency;

when taking any technical decisions all safety rules should be complied with and the necessary sanitary and hygienic conditions provided.

In practice, several different solutions meeting the safety requirements can usually be found, therefore the final choice should take into account these considerations, primarily the economic ones.

The basic calculation methods are: *statistical*, *observational*, *cut-and-try*, *mathematico-analytical*, *grapho-analytical* and the *method of variants.*

Statistical method. By mathematical processing of statistical information one can establish quantitative relationships. Using individual points corresponding to initial numerical values and plotting them in Cartesian coordinates it is possible to construct empiric probability curves and one can find mathematical equations to match them.

Method of observations. When statistical data are not available or insufficient the initial numerical values can be obtained by carrying out appropriate observations.

Experimental (cut-and-try) method. Inasmuch as direct observations (for example those of technological operations) are often conducted under conditions which are not quite comparable, experiments can be made in order to establish a particular relationship with a greater degree of precision. In carrying out experiments the effect of factors other than those under study should be more or less equal.

Mathematico-analytical method. [1] In this procedure the unknown relationship is expressed as a mathematical formula. Using the formula and known numerical values, charts or numerical tables can be obtained. To solve some problems of mining engineering, the optimal values of certain magnitudes have to be determined. In such cases use is made of mathematico-analytical techniques based on establishing a functional mathematical dependence between the value sought for and the resultant cost.

Let us assume that $S = f(x)$.

The most rational magnitude of the desired value should correspond to the extremal magnitude of S, in other words the mathematical solution amounts to finding the maximum or minimum of the function by means of higher mathematics.

According to the theory of maxima and minima, taking the first derivative and equating it to zero we arrive at the equation

$$\frac{dS}{dx} = 0,$$

from which the desired value can be found. The analytical method can be employed in solving a number of problems associated with the mining of mineral deposits, viz. determining the mine-field size, the level interval height, the depth of open pits, the height of a bench, etc.

Solutions obtained by using the analytical method should be correlated with practical results of similar enterprises. In solving problems by the analytical method account should be taken of the property of the function under study to vary only slightly in the region of its maximum and minimum with appreciable changes in the independent variable, i.e. in actual practice one should keep in mind an entire range of optimal magnitudes rather than a single value of the desired magnitude.

This allows the ultimate decision to be taken with due consideration for a number of other mining conditions and without incurring any substantial economic loss. While using the analytical method one should not excessively complicate the calculations, nor unduly simplify them.

A series of technical and economic figures should be utilized, although with great caution. In the absence of reliable figures suitable for the specific conditions of the problem in hand it is advisable to obtain them through overall estimation or detailed calculation.

Grapho-analytical method. Since the solution of certain problems of mining engineering by using known mathematical procedures

[1] This method is often called analytical.

is a matter of extreme complexity it should be done graphically, on the basis of established mathematical relationships.

Method of comparing variants. The extremely large number of geological and mining factors intractable mathematically and often varying in value, which must be considered in solving problems of early development, complicates and frequently rules out the use of the higher mathematics procedures, e.g. establishing of functional dependence and finding the maximum or minimum of the function. Therefore, in tackling many problems of mining bedded deposits and, particularly, ore bodies it is habitual to apply yet another method—that of comparing variants.

When employing this method one assumes several values for the sought magnitude and calculates the resultant cost for each of them. The results of these calculations may be tabulated or presented in the shape of a graph. When drawing a graph the sought magnitudes are usually marked off on the x-axis and the corresponding cost results on the y-axis.

By connecting the points representing the cost in the different variants a curve characterizing the cost variation and the region of its minimal values can be obtained. The accuracy of the curve depends upon the number of points. This method is widely used in choosing the method of early development, mining, etc.

Below we give basic information on the opening-up of bedded deposits. The examples of early development are mainly derived from experience gained in mining coal deposits of the USSR, which as a rule have regular mode of occurrence. When studying problems of mining sheet deposits one should refer to the definitions set forth in the regulations for working coal deposits in the USSR as regards the classification of seams according to their thickness and angle of dip.

As far as their thickness is concerned, the bedded deposits are subdivided into very thin (up to 0.5 m), thin (from 0.5 to 1.3 m), medium (from 1.3 up to 3.5 m) and thick deposits (over 3.5 m).

With respect to their angle of dip the beds are classified as gently dipping (with an angle of dip up to 25°), inclined (from 25 to 45°), and steeply dipping beds (over 45°). Beds with an angle of dip less than 5° are conventionally referred to as flat, or level beds.

2-2. DETERMINATION OF ANNUAL OUTPUT, SERVICE LIFE AND LEVEL INTERVAL HEIGHT

The basic value in projecting a mine is the amount of mineral resources. Work on coal mine projects can only be started if the available resources of the categories A_2 and B make up not less than 50-60% of the sum-total of the categories $A_2 + B + C_1$.

In estimating the service life of a mine we take into account 100% of inventory reserves of the categories $A_2 + B$ and 50% of the category C_1; to arrive at the workable reserves, we deduct the planned overall losses of coal—the coal abandoned in pillars underneath protected structures, the coal near the boundaries of mine-fields and the operational losses of coal.

In projecting mines one must bear in mind that an increase of their planned production capacity brings with it rising labour efficiency, reduced production costs and lower capital investments per ton of mineral obtained.

In the coal industry of the USSR the existing trend is towards construction of large-sized mines and amalgamation of low-output mines into big integrated enterprises with centralized surface and, in some cases, underground facilities.

With particularly favourable mining and geological conditions and in areas with extremely rich coal reserves it is advisable to project mines with a daily output of up to 20,000 tons.

Depending on their production capacity the service life of mines should be adopted as follows:

Output, tons/day	Service life, years
1,500	Not less than 30
2,000-3,000	ditto 40
Above 4,000	ditto 50

The latest approved projects for mines of the Donets and Kuznetsk basins envisage service life periods of up to 75-80 years for a number of high-productivity mines. Long service life periods up to 80-100 years are adopted for steeply pitching seams opened up by sinking deep vertical shafts.

Calculations show that these periods are in keeping with economic demands, since the total initial and running expenses per ton of coal (which depend on the mine output) will then be reduced to a minimum.

In areas with limited amounts of useful mineral it is expedient to construct mines with an output of 1,000 tons per day or less and with a service life of 10-15 years, provided the surface structures have simplified design and are made of light pre-fabricated members. There are also mines of lower production capacity, productive-exploratory mines with an output of up to 200-300 tons of coal per day.

For large-sized sheet deposits in the USSR the mine output is usually not calculated, but established in terms of planned figures contingent upon the demands of the national economy for the particular useful mineral, with subsequent verification against the possible conditions of actual mining.

Checking the Annual Mine Output and Determining the Level Interval Height by the Analytical Method

The production capacity of a mine can be estimated on the basis of the practical or technically admissible height of the level interval and other conditions of mining (the level-wise method of preliminary development is the principal technique in the preparation of mine-fields). The reverse procedure is also possible, i.e. the determination of the level interval height from a pre-set annual output of the mine and the technical conditions of mining. In working gently dipping and inclined strata we usually determine the level interval height from the pre-assigned production capacity of the mine; for steeply pitching beds the annual output of the mine is estimated from the level interval height chosen on safety grounds. The formula given below allows us to determine the annual output of a mine or the level interval height in working a single bed:

$$A = 2LhpC$$

where A = annual output of the mine, tons;
2 = number of wings, or flanks in the level; in sheet deposits both flanks are often worked simultaneously;
L = average advance of stoping operations along the strike in a wing of one level per year. The value of L usually depends on the adopted method of mining, equipment and organization. In working coal deposits of the USSR L reaches 600 metres or more per year (the standard rate being 480 m). The annual advance of stoping can be measured more accurately in each individual case by adopting the following set of figures: 300 workdays per year, one or two cycles per day with wide-range (or web) extraction (1.5-2.0 m) and two-four cycles with narrow-range (web) extraction (0.6-1 m), and a coefficient of 0.85-0.90 for the geological conditions in the mine;
h = inclined height of level interval, m;
p = output per 1 sq m of the bed, tons/m^2;
C = coefficient of recovery from the deposit, which usually varies from 0.9 to 0.8, depending on the adopted mining method and the conditions of stoping operations.

The level interval height is made up of the length of the corresponding stoping face lines, the aggregate height of the pillars abandoned in the level and the total width of the development workings (drifts, crossheadings).

In determining the annual output of mines with very extensive fields account should be taken of the possible increase in the aggre-

gate productive faces in each flank of the level, which can eventually form two active stope lines pushed forward from the centre of the wing in both directions, i.e. towards the shaft and the mine-field boundary along the strike.

The above formula is based on the condition of ensuring the annual output of a mine by working one level.

Considering the possible diversity of conditions in mining bedded deposits (simultaneous extraction of several seams and levels, varying number of wings in the level), the annual output or the inclined level interval height is estimated by the formula:

$$A = n_w n_e k_{se} \Sigma p L h C$$

where n_w = number of simultaneously worked wings in the mine-field; it varies between one and four (it is more than two in folded series of active beds);

n_e = number of levels worked at a time; simultaneous extraction in several levels results in increased maintenance and service expenses per level, and should only be practised as an exception in sheet deposits of extremely limited extent;

k_{se} = coefficient of simultaneous extraction of beds, i.e. a figure indicating what part of the total productivity of the beds in the series is, on the average, being actually utilized simultaneously; in some cases simul taneous working of several seams is inadmissible because of the possible undermining of some sections by others; in such instances some sections should be mined in advance of others; in mining a series composed of many seams no more than three-four sloping or inclined beds and 10-12 steep ones should be worked simultaneously;

Σp = total productivity per 1 sq m of beds mined in the series, tons/m^2;

L = average annual advance of stoping in one wing of the mine-field, m;

h = inclined height of level interval, m;

C = coefficient of recovery.

For technical and safety reasons, in steeply pitching beds the inclined height of the level interval should not exceed the limits established by experience. It is generally set at 120-150 m for thin and medium seams, and 70-100 m for thick ones. In gently sloping beds the height may vary within a wide range (from 300 to 600 m, more often between 300 and 400 m); it is fixed in conformity with the pre-determined mine output, due account being taken of the technical means available for mechanization of operations.

2-3. ESTIMATING THE SIZE OF A MINE-FIELD AND THE SEQUENCE OF ITS WORKING

Estimating the Mine-Field Size

It has already been mentioned that in mining sheet deposits with very large reserves the projected annual output figures are usually subsequently checked with relation to the actual possibilities.

Once the yearly production capacity of a mine is fixed, there are definite dimensions of the field along the strike and dip at which the cost of production, early development and operations (contingent upon the method of mine opening) per 1 ton of extracted mineral will be at its lowest.

It should be borne in mind that one part of the expenditures per 1 ton of output will be the higher the larger the mine-field (for example, haulage along the drifts and their maintenance). Another part will, on the contrary, decrease with the extension of the mine-field (for instance, depreciation of capital investments for the erection of surface structures). And, finally, the third part of the expenditures is independent of the size of the mine-field (cost of stoping).

The length of the mine-field along the strike augments with the rise in the annual output. Thus, according to the calculations made in working out the projects for new mines in the Donets basin the length of mine-fields is set at 2,500 to 5,400 m (averaging 4,800 m), in the Karaganda basin at 3,500-6,000 m (averaging 4,600 m), and in the Kuznetsk basin at an average of 6,100 m. The smaller figures refer to mines with an annual output from 200 to 600 thousand tons, the larger are for 1,200 to 2,000 thousand tons. The size of mine-fields on the pitch in the case of gently dipping and inclined beds is taken to be 40-60% of their length on the strike, while in steeply pitching strata it reaches 1,000-1,200 m in the vertical direction. With the present-day methods of transportation and the current cost of maintaining workings which are as a rule supported by reinforced concrete and steel members, it is technically and economically expedient to have mine-fields extending 10-12 and more thousand metres along the strike (provided the deposit has appropriate reserves).

In recent years coal mines with mine-fields exceeding 5-6 km in length have been built in the USSR, for example, the mine "Oktyabrskaya" in the Donets basin has a mine-field measuring 7.5 km on the strike and 4.4 km on the dip.

If reserves of a folded deposit are not divided into individual isolated sections by geological disturbances and it has a regular mode of occurrence and very large dimensions, the size of its mine-field can be estimated by the mathematico-analytical method. The independent variables are the length of the field along the stri-

ke, L, and the number of levels on the dip, n, with a definite interval height. Here, the cost factor (sum-total of capital outlays and running expenses per 1 ton of the output) is a function of several variables, i.e.:

$$\Sigma Q = f(L, n)$$

Taking partial derivatives and equating each of them to zero we arrive at two equations:

$$\frac{d\Sigma Q}{dL} = 0$$

$$\frac{d\Sigma Q}{dn} = 0$$

Simultaneous solution of these equations yields the sought values for L and n. The number of levels allows us to define the size of the mine-field along the strike. The system of equations with two

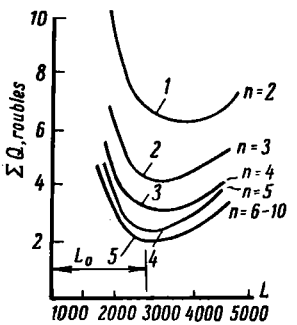


Fig. 8. Charts for estimating the size of a mine-field:

n—number of levels; *L*—size of the mine-field along the strike, m; *1*, *2*, *3*, *4*, *5*—curves illustrating changes in the expenditures per 1 ton of workable reserves

unknowns can be solved graphically. To do this, figures 1, 2, 3, etc. are substituted for n in the equation with L, and 200, 400, 600 metres, etc. for L in the equation with n. The respective values of L and n are plotted on a chart. The optimal value of L_0, n_0 is found at the intersection point of curves L and n.

As has already been noted, it is not the point that is important, but the region of optimal values, which may have wide limits, considering the possible inaccuracy of the initial data. The optimal values for L_0 and n_0 estimated on the basis of minimal expenditures are merely tentative and should be defined more precisely with due account taken of a number of other factors. A detailed estimation of the size of a mine-field by the mathematico-analytical method is given in works of Academician L. Shevyakov. One of the principal conditions this method has to satisfy is the correctness of the cost figures used in the calculation.

The most advantageous size of a mine-field can also be found by means of technical and economic estimates with subsequent graphic representation of the results obtained (Fig. 8). For this purpose one has to assume a consecutive number of levels (2, 3, 4, etc.) within the range of interest and each time make an estimate of all accountable expenses ΣQ per 1 ton of workable reserves for a number of values of the length of the mine-field L. The results are used for plotting graphs.

By studying the graphs we solve the problem posed. In our example expenses per 1 ton of reserves with the levels numbering from 6 to 10 vary so slightly that the adopted scale of the drawing makes it difficult to trace the curves. This points to a broad range of optimal values.

For deposits having an irregular shape or irregular mode of occurrence the size of the mine-field is, as a rule, defined by the method of comparing variants.

Sequence of Working a Mine-Field

Sequence of Working Levels in a Mine-Field. To cut down the time needed for prospecting and shaft sinking the levels are ordinarily mined in descending order, except for the portion of the mine-field extending along the rise in a gently dipping bed, in which case one should compare the descending and ascending orders of mining the levels and choose the more suitable method.

Number of Wings (Flanks) in a Level. All other conditions being equal, a bilateral mine-field is to be preferred to a unilateral one because of the lower cost of haulage in the drifts and their maintenance and the reduced ventilation expenditure.

As regards mine aeration, the best practice is to divide the mine-field into two flanks with equal amounts of mineral reserves, using the unilateral alternative only in the event of exceptional, compelling conditions of mining. The number of wings in the level should accord with that in the mine-field as a whole.

Sequence of Working a Level. Each level in the mine-field may be worked in the direction from the hoisting shaft towards the mine-field boundaries (advance mining), or from the mine-field boundaries towards the hoisting shaft (retreat mining) (see Fig. 9).

The principal advantage of the first method is that it allows quick deployment of stoping operations. But the method has serious shortcomings, viz. unfeasibility of early exploitation of the level or of its producing sections, which complicates the stoping operations and may cause a temporary stoppage of mining in the working faces; difficulty in maintaining the entries, subdrifts and, especially, airways in the worked-out portion of the level; high losses of coal

in robbing the pillars owing to their frequent crushing; large leakage of air through mined-out spaces; increased hazards of water or quicksand inrushes and difficulty of controlling them; great danger of spontaneous combustion of coal due to increased losses of coal

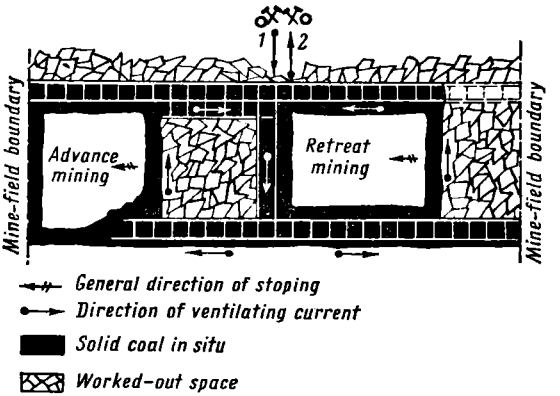


Fig. 9. Advance and retreat mining in a level with shafts sited centrally: *1*—hoisting shaft; *2*—ventilating shaft

and air leakages through the worked-out spaces; complexity of transportation in the haulage entry, since the extracted masses of rock and mineral are delivered simultaneously from active working and heading faces.

The above-listed shortcomings of advance mining are obviated completely or partially in working a series of beds with drivage

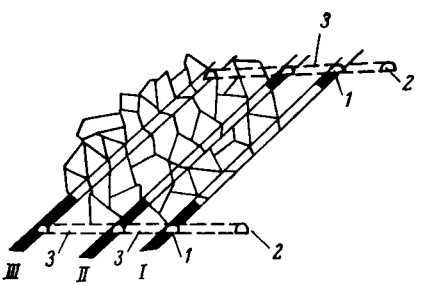


Fig. 10. Disposition of gathering (group) and lateral entries: *I*, *II* and *III*—coal seams; *1*—group entries; *2*—lateral entries; *3*—cross-entries

of gathering (mother, group) or lateral drifts and also in working seams less than 0.8 m thick, where the integrity of entries is only slightly affected by stoping.

In mining a series of beds (Fig. 10) main drifts of a group of closely lying seams may be connected by intermediate or auxiliary crosscuts so that transport, passage of men and the flow of main air

currents are concentrated in the drifts of the lower bed in the group. Such drifts are termed group or gathering (mother) drifts. Should it prove difficult to maintain them, they are cut in the country rock at some distance from the lower seam. Wherever a gathering or a level drift exists the corresponding sections of entries driven in beds may be abandoned as the working faces are pushed forward.

The disadvantages of advance mining are either absent or considerably reduced in retreat mining, but working of a level from the boundaries of the mine-field towards the hoisting shaft has shortcomings of its own. These include: extra time needed to develop the level before its actual exploitation can be started; difficulty of providing ventilation while driving long drifts; large initial outlays for development work.

Modern ventilation facilities enable drifts with cross-entries to be extended over considerable lengths.

The most serious of the disadvantages outlined is the delay in preparing the level for exploitation. This causes difficulties in developing the first level; the subjacent levels can be prepared in good time. The time of level development can be reduced by high-speed drivage of roadways, bringing the average monthly advance rate up to 150-200 m and more. In the USSR mining combines (heading machines) ПК-3 (ПК-3м) and others are widely employed in drivage of mine openings. The average advance rate of headings accomplished with the aid of the ПК-3 (ПК-3м) combines reaches 170 m per month. This combine can extract coal and rock separately.

In individual instances the levels can be mined by the retreat method only under conditions of extremely high pressure of wall rocks and serious hazards of spontaneous combustion of coal.

The above comparison of the advance and retreat orders of mining horizons refers to hoisting and ventilating shafts located in the centre of the mine-field. With a ventilating shaft sited near the upper boundary of a mine-field a number of the cited advantages of retreat mining are lost owing to the need of maintaining drifts in the mined-out section to ensure proper ventilation.

In choosing one or the other sequence of working in the level, account should be taken of the local geological and technical conditions of mining.

Today, new rock-tunnelling machines and equipment have been designed and widely introduced, this enabling mine openings to be pushed forward at a high speed. Therefore it becomes possible in many instances to change over from the advance to the retreat method of mining.

Being the more effective, retreat mining should be adopted as a principal method in working levels and production faces in the panels of collieries.

2-4. TYPICAL METHODS OF OPENING-UP SINGLE-SEAM DEPOSITS

Under certain conditions of mining bedded deposits use is made of specific methods of early development, whose suitability is usually supported by technical and economic calculations. Knowing the typical methods of opening-up mineral deposits one can select the most expedient techniques and, after comparing the technical and economic data, make the final choice of this or that method. Below we describe typical methods for opening-up a single seam under a definite set of mining conditions.

Diagrams illustrating early development with special reference to a particular method only indicate the location of the main shaft across the strike.

Information as to the disposition of other shafts is presented later, following the description of the most common methods of early development.

Opening-Up a Gently Dipping Bed Through a Vertical Shaft with a Permanent Incline and a Slope

A typical way of opening-up a gently dipping seam is through a vertical shaft with a permanent incline and a slope (Fig. 11). With this method the vertical shaft is sunk on the line marking the

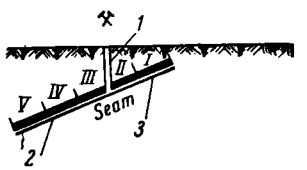


Fig. 11. Opening-up a gently dipping seam:
1—vertical shaft; *2*—slope; *3*—brake incline; *I*, *II*, *III*, *IV* and *V*—levels

dip of the seam so as to provide for the extraction of the up-dip mine-field through a permanent incline and of its down-dip portion through a permanent slope. The rational length of the slope and incline is determined by technical and economic estimates. In choosing the length of the permanent slope account should be taken of difficulties in ventilation and mine drainage when working the lower levels of the mine-field. In high-output mines it may prove expedient to have, not just one, but two or even several permanent inclines or slopes within a mine-field. A permanent incline or slope is one servicing several levels.

Opening-Up a Steeply Pitching Seam Through a Vertical Shaft with Crosscuts

A typical method of opening-up a steeply pitching bed is through a vertical shaft with crosscuts. The shaft may be sunk in the foot wall, the hanging wall, or it may intersect the pay bed (Fig. 12).

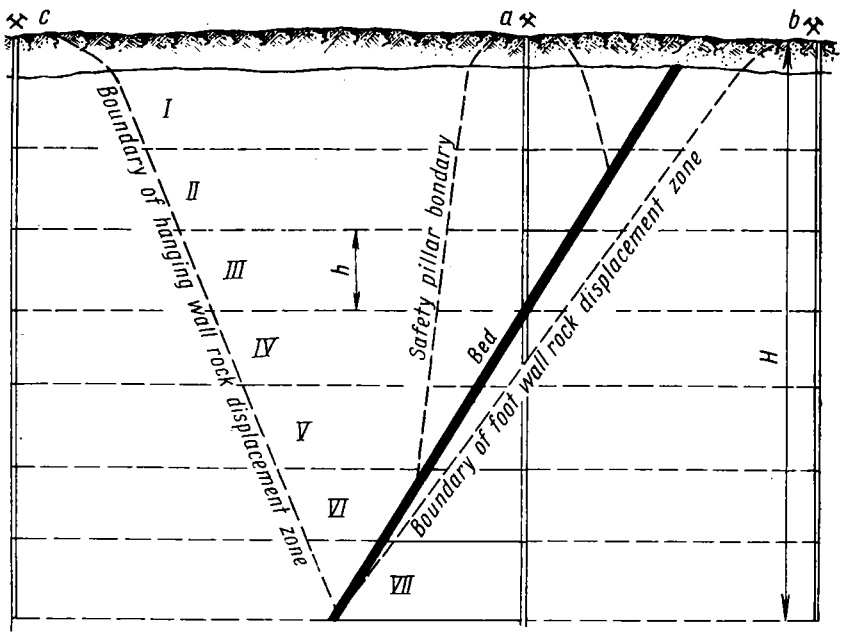


Fig. 12. Possible dispositions of the main shaft in opening-up a steeply pitching bed:

a—position of the shaft when it strikes the pay bed; *b*—position of the shaft in the foot wall beyond displacement zone of the foot wall rocks; *c*—position of the shaft in the hanging wall beyond displacement zone of the hanging wall rocks; *H*—shaft depth; *h*—level interval; *I-VII*—levels

In the latter case the crosscuts will have a minimum extent which permits appreciable reduction in the cost of excavation and maintenance of these workings, as well as cutting down expenses for hauling loads along them. The drawback is the necessity of leaving a protective shaft pillar.

In working thick high-dipping seams their upper portions may be mined by the open-cut method, while the lower ones are worked by the underground method, usually through a shaft sunk in the hanging wall beyond the rock displacement zone.

It should be pointed out that although protective pillars are robbed prior to closing down of the mine, they nearly always entail economic losses, since no more than 75-80% of the mineral is ordinarily recovered from them. Relatively few cases of complete recovery of protective pillars are known and this always involves costly methods of extraction. Because of increased rock pressure the recovery of the mineral from a protective pillar is more expensive than mining by stoping faces.

Losses due to protective pillars are especially great in mining valuable minerals under adverse conditions of pillar robbing and in deposits containing insignificant reserves of the useful mineral.

In mining coal deposits with extensive mine-fields and thin seams it is common practice to leave protective shaft pillars. Then the reserves in the pillar are of limited proportions and can be recovered with lower losses.

Shaft disposition in the hanging wall is less advantageous because of the great length of crosscuts. It is tolerated only in specific mining conditions when the shaft cannot be located elsewhere.

Opening-Up an Inclined Bed Through a Vertical or an Inclined Shaft

Typical methods of opening a tilting seam include early development through a vertical shaft with crosscuts, a gravity incline and a slope (Fig. 13), and through an inclined shaft (Fig. 14).

The first of the above-mentioned procedures is essentially a combination of methods employed in opening high-dipping and slightly

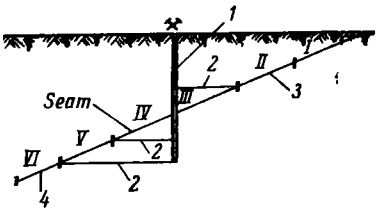


Fig. 13. Opening-up an inclined bed through a vertical shaft with crosscuts, gravity incline and slope:
1—shaft; *2*—crosscut; *3*—gravity incline; *4*—slope; *I*, *II*, *III*, *IV*, *V* and *VI*—levels

pitching seams. Here, crosscuts are only driven in some of the levels. The number of horizons to be opened directly through crosscuts is determined in each individual case by technical and economic calculations.

In opening-up a bed through an inclined shaft this is most often sunk in the mineral bed itself. With an irregular mode of occurrence or when the slope angle of the shaft has to be decreased, for instance with conveyer transport provided along the shaft, the shaft is driven in the foot wall rocks at the appropriate distance from the seam.

Level entries are driven from the inclined shaft in each horizon, or short crosscuts are excavated if the shaft passes through the foot wall rocks.

The advantages of opening-up through an inclined shaft include the following: the possibility of verifying the information gained during exploration of the mineral bed; while driving the shaft the data on the structure and properties of the bed and wall rocks are supplemented; the extent of the weathering zone is established; extraction of the useful mineral during shaft sinking is possible;

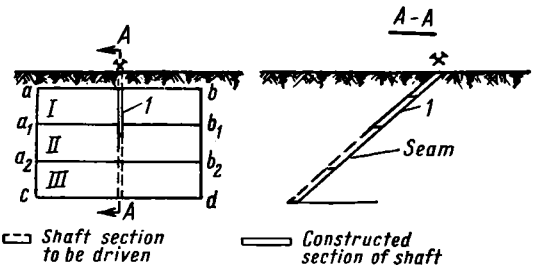


Fig. 14. Opening-up a seam through an inclined shaft driven in a bedded mineral deposit:

1—shaft; *a-b-c-d*—mine-field; *a-b*; a_1-b_1, a_2-b_2, *c-d*—level drifts; *I*, *II*, *III*—levels

the shaft can be sunk without intersecting aquifers overlaying the mineral bed; the possibility of setting up a highly efficient hoisting belt conveyer.

Among the drawbacks of an inclined shaft as compared to a vertical one are: the greater length of the shaft and the lower permissible hoisting speed; the increased pressure of wall rocks; the larger cross-section of shafts with cage hoisting.

Notwithstanding these shortcomings opening-up through an inclined shaft is practised in a number of coal basins and districts. Inclined shafts are used particularly often to deliver the mined bulk from the faces to the ground surface by belt conveyers. In this case the inclined shaft forms one of the links in an all-round conveyer line (Karaganda coal basin). Opening-up through an inclined shaft sunk in a mineral bed is favoured by a small thickness of the overburden or an outcrop of the seam to the ground surface, regular occurrence and a persistent angle of dip, the absence of the threat of self-ignition of the coal.

The great strides made in the development of conveyer transport and the advanced designs of the extensive belt-conveyer lines adapted to operate at various gradients allow inclined shafts to be employed more widely in opening-up of mineral deposits.

Opening-Up a Horizontal Bed Through a Vertical Shaft

A typical method of opening a horizontal bed or a seam with a very slight angle of dip is through a vertical shaft sunk in the centre of a mine-field with provision made for a protective pillar (with a bed of small thickness and a large-sized mine-field), or through a shaft driven beyond the boundaries of the mine-field, outside the zone of rock displacement (Fig. 15).

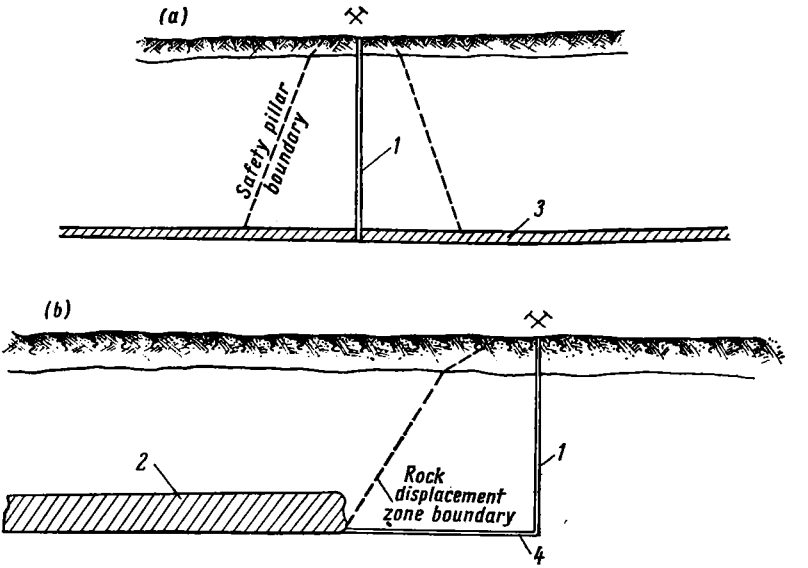


Fig. 15. Opening-up a flat bed through a vertical shaft (side elevation):
a—intersecting the bed; *b*—sunk beyond the enclosing rocks displacement zone; *1*—shaf t
2—thick bed; *3*—thin bed; *4*—lateral entry

In opening-up a flat or a very slightly inclined seam one can choose the number of producing panels and establish the order and direction of their extraction. It is advisable to arrange the hauling of loaded mine cars towards the shaft on the downgrade and create favourable conditions for the runoff of mine water.

In solving problems associated with the opening-up of flat seams relief plans of the bottom and roof of the bed in contour lines (hypsometry) are utilized along with vertical geological sections.

The final choice of disposition of panels is made by comparing the possible versions from the technical and economic standpoints.

Inclined shafts are also used in opening-up horizontal beds occurring at a small depth. They simplify the surface structures layout and enable the use of conveyer installations.

Examples of flat or nearly flat (up to 3°) deposits are the coal occurrences in the Moscow and Cheremkhovo basins, and also many pyroschist deposits in the USSR.

Opening-Up a Bed Through an Adit

A bed in a mountainous region can be opened by an adit (Fig. 16). As compared to opening-up through shafts this method has the following important advantages: pumping expenditures are eliminated,

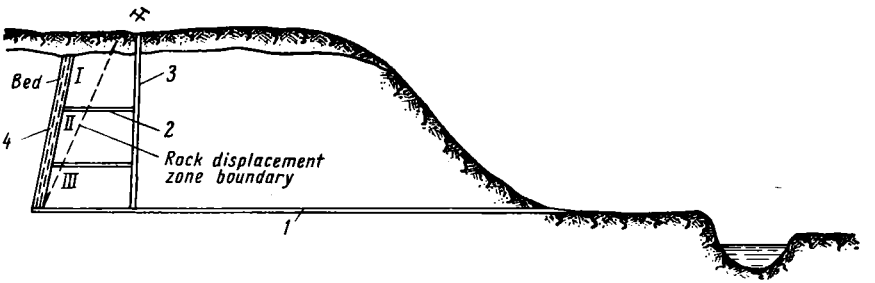


Fig. 16. Opening-up a seam through an adit:
1—adit; *2*—crosscut; *3*—service shaft; *4*—coal chute

except for insignificant amounts spent on the upkeep of drain ditches; the outlays for building surface structures and plants are considerably reduced; there is no need for a headframe, shaft- and hoist-engine houses; the adit can be pushed forward at a greater rate by comparison with shaft sinking; much time is saved which would have been lost in early development through shafts in driving crosscuts and excavating shaft station workings and nearby service rooms; no expenses on hoisting are incurred; haulage in the adit is done conveniently and cheaply by powerful electric locomotives, with the mineral being delivered directly to a storage place or a railroad bin.

For these reasons adits should be used wherever conditions of mining favour their drivage and exploitation.

An adit may be driven across the strike, along its line, or diagonally to it. In opening-up through an adit the slope angle of beds may differ widely if the regional topography and geological position of the deposit permit it, the horizon of the adit should be as low as possible, so that maximum possible reserves of the mineral could be undermined. Short level adits, or an auxiliary shaft are driven to secure communication between horizons and to be able to deliver supplies and equipment to each of them. Depending upon the angle of pitch of the seam the extracted mineral is transferred to the main adit along gravity inclines, slopes, or through special chutes.

When choosing the site and elevation mark of an adit the following conditions should be complied with: the adit mouth is to lie above the level of a possible rise of water in neighbouring impounded bodies; the site near the mouth of the adit must be fairly level and large enough to accommodate the necessary surface structures, spur-lines, bins and storage places for the mineral, and waste dumps. The mineral should pass through bins and beneficiation installations of the concentration plant by gravity. The area at the mouth of the adit should be protected from rock falls and snow slides.

Opening sheet deposits by an adit has been done in many Soviet coal mines.

2-5. STANDARD METHODS OF OPENING-UP COAL MEASURES

Depending upon different factors a coal measure may be opened up by using various methods (Fig. 17), viz. through individual shafts (Fig. 17*a*), through a vertical shaft with a permanent crosscut (Fig. 17*b*), through a vertical shaft with level crosscuts (Fig. 17*c*), through a vertical shaft with a permanent winze (Fig. 17*d*), through an inclined shaft with level crosscuts (Fig. 17*e*), and—with local topography favouring it—through an adit (Fig. 17*f*). In addition to these simple ways of opening-up a coal formation, use can be made of combined ones, as shown in Fig. 17*g* and *h*.

The choice of the method of opening-up is influenced by the following factors: mode of occurrence of the deposit: its angle of dip, number of seams and their spacing, thickness of the seams, hardness of enclosing rocks; the proposed mining depth; the planned production capacity of the mine; prevailing local conditions; the suitability of the surface topography for locating shafts and associated structures and plants; the size of areas allotted for mining, etc.

In heavily pitching deposits it is expedient to open coal measures through a vertical shaft with one permanent or a few level crosscuts (Fig. 17*b*, *c*). In the case of a vertical shaft with a permanent crosscut each seam is opened individually through permanent or panel gravity inclines and slopes.

In opening-up gently dipping deposits it is practical to use a vertical shaft with a permanent winze (Fig. 17*d*). With this method the mineral passes from upper seams through the winze down to the bottom station level, wherefrom the mineral is hoisted up. Each seam is mined separately and is provided with a permanent gravity incline and a slope of its own.

As the number of seams increases the cost of opening-up the deposit through individual shafts for each seam also rises, whereas the cost of opening-up through one shaft diminishes, since the capital outlays are distributed largerover tonnages of the developed reserves.

Wide spacing of seams favours opening-up each of them through a separate shaft (Fig. 17*a*). In this case sinking a single shaft for the entire coal formation, which would involve excavation of extensive crosscuts or winzes, may require greater expenditure than

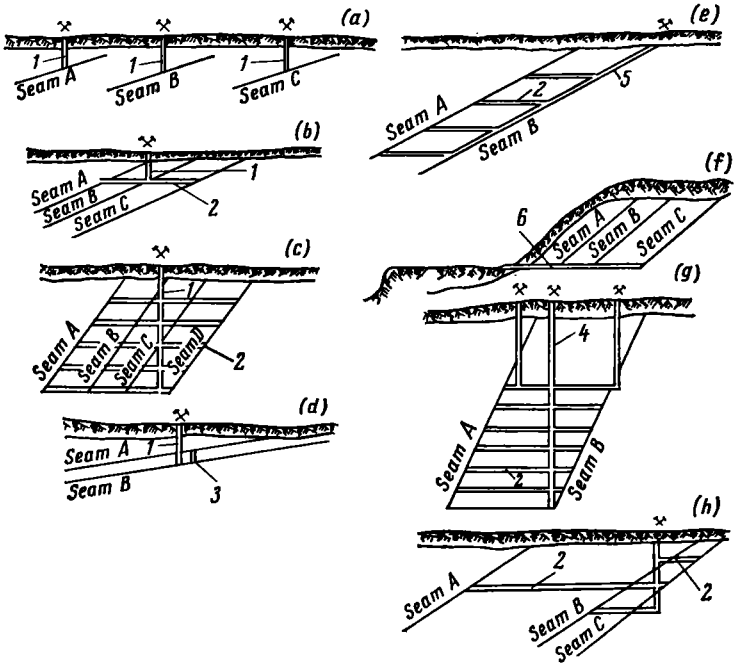


Fig. 17. Early development of a series of formations:
a-f—simple standard methods; *g*, *h*—combined methods of opening; *1*—vertical shaft; *2*—crosscut; *3*—permanent winze; *4*—central vertical shaft; *5*—inclined shaft; *6*—adit

sinking and equipping separate shafts. Conversely, close spacing of the seams in the measure favours their joint development through one central shaft.

The advantages of opening-up a coal formation through one central shaft also augment with increased depth of mining, since in this case it is not always practical to sink a separate shaft in each seam down to a considerable depth. In mining deep-lying levels through a central shaft it may, however, prove economical to open up upper levels in a coal measure through separate shafts (Fig. 17*g*).

This is in fact a combined procedure involving early development through a central and separate shafts. Figure 17*h* shows a combined method of opening-up through a vertical shaft with a permanent and level crosscuts when one of the seams lies at an appreciable

distance from two others. In different mining conditions other combined methods of opening-up coal measures may also be applied.

In circumstances favourable for driving inclined shafts this method may be employed for early development of two or several seams (Fig. 17*e*). To reduce coal losses in the protective pillars the inclined shaft is sunk through the lower seam of the measure. When an inclined shaft is used for opening-up the deposit the seams are connected by one or several crosscuts, depending upon the angle of dip.

When deciding upon the production capacity of an enterprise and time-schedule for its commissioning substantial corrections

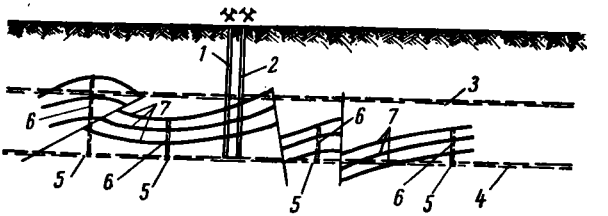


Fig. 18. Opening-up a strongly dislocated deposit:
1—hoisting shaft; *2*—ventilating shaft; *3*—ventilating crosscut; *4*—haulage crosscut; *5*—lateral entry; *6*—winze; *7*—coal beds

in the choice of the method of early development may be introduced. For example, if, with a given mode of occurrence, it appears most advantageous to open the deposit through a deep central shaft whose sinking demands much time, while the actual extraction of the mineral is expected to start at short notice, it is advisable to begin mining through separate shallow shafts, thus reducing the time needed for opening and developing the deposit. The additional expenses caused by adopting a less economical method of opening may be offset by the advantages of quick delivery of a badly needed mineral.

In conclusion it should be emphasized that the more intricate the shape of a deposit, the more complex is the method of its opening, particularly in mining beds with disturbed mode of occurrence in the presence of shifts and faults. Individual portions of such a deposit are opened up in a variety of ways, depending upon the geological and mining conditions prevalent in its particular sections. For instance, in a gently dipping coal-saturated series (Fig. 18) the foot wall is pierced by the main haulage crosscut with a large number of lateral entries and winzes of differing height. There are also many other modes of early development. It is especially important in planning the opening of intricately shaped deposits to obtain

accurate geological data in good time. In this case the geological and mining characteristics of individual portions and sections of the deposit should be determined and shifts, faults and other disturbances located.

2-6. FINAL CHOICE OF A METHOD FOR OPENING-UP A DEPOSIT

In the foregoing, standard and also combined methods of opening-up seams or strata of seams were listed. In many instances the choice of a method is difficult because of the possibility of applying several, often two or three alternative methods of early development. It is particularly hard to select a combined method for opening-up an intricately shaped deposit. In all such cases one should compare practical procedures for early development after performing technical and economic calculations. Below we set forth the pattern for calculations along with the rules to be followed in applying the widely used method of comparing variants.

Proceeding from a definite plan for opening a deposit, capital outlays and operating expenses, as well as expenses for repairing and maintenance of mine workings are calculated, following which other practical versions of early development (with some diversions from the first one) are considered and the relevant expenditures determined. By comparing the aggregate expenditures estimated for each version the optimal alternative is found which is then used in opening-up the deposit. In making the final choice consideration should be given to the convenience of exploitation, the initial capital outlays, the prospects for increasing the output and the dates for the commencement of stoping operations. The lastly named factors may prove decisive in certain cases.

When comparing the variants, the following basic rules should be complied with: compare alternatives of the highest practical value; include in the calculation only the most essential items of expenditures in rounded off figures; disregard cost items similar in size for the versions under consideration; refer the expenses to one and the same period of time, preferably to the entire period of mining the deposit; when the difference in cost is less than 10% the alternatives should be considered economically equivalent, the version most convenient from the technical viewpoint being then adopted; in estimating cost items attention should be focused on the payroll, which usually constitutes the bulk of the expenditure.

The stipulated pay-off periods should be taken into account in estimating various capital outlays and running expenses. For mining enterprises this time may be taken as 10 years, this corresponding to a standard (annual) efficiency factor of 0.1. With due

account taken of these points the variants should be compared according to the formulas:

$$C_1 + EK_1$$

$$C_2 + EK_2$$

where C_1, C_2 = per ton mining cost for each variant, roubles;
E = standard efficiency factor;
K_1, K_2 = per unit capital investments for each variant, roubles/ton.

Per unit capital investments are determined by dividing the aggregate sum of capital expenditure for the construction of the mine by its annual output.

Adopt the variant where the $C + EK$ value is the lowest.

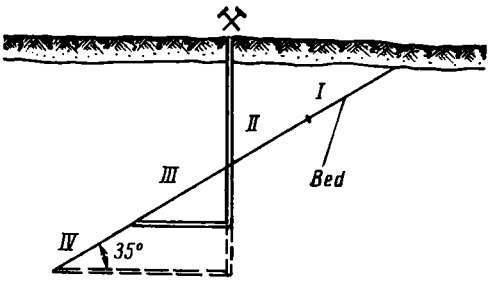


Fig. 19. Choosing method for opening-up the bottom level:
I, *II*, *III* and *IV*—levels

If the construction is to be completed within different time periods the capital outlays in each variant are estimated with due regard for

$$K_{prod} = (1 + E)^t$$

where E = standard efficiency factor;
t = construction period (capital outlays for each variant to be multiplied by K_{prod}).

Below we give an example of solving a particular problem by the method of comparing variants: the choice of a method for opening-up the last (fourth) level of a mine-field in an inclined bed with the shaft sunk down to the elevation mark of the third level (Fig. 19). With an easy dip of the seam opening-up of the lower level may be carried out through a slope (first variant) or a crosscut (second variant). In the latter instance a preliminary deepening of the shaft down to the elevation mark of the last level is required, which should be taken into account in making the comparison. Table 1 compares expenditures for two variants of early development under the following conditions of mining: annual output of the mine—250 thousand tons, time limit for mining the level—5 years, inclined level interval height—200 m, workable coal reserves in the level—1,250

Table 1

Comparison of Expenditures for Different Variants

Items of expenditure	Expenditures, thous. roubles	
	Variant I	Variant II
Excavation of workings and openings		
Shaft	—	20
Shaft station	—	20
Crosscut	—	5
Slope with manway and other openings	21	—
Slope winder chamber	1	—
Maintenance of workings		
Slope with manway	2.5	—
Crosscut	—	0.5
Hoisting		
Labour cost at shaft station	equal	
Labour cost at slope	20	—
Mechanical equipment and plant of slope	3	—
Electric power	equal	
Mine drainage		
Labour cost for auxiliary mine pumping	3	—
Electric power	equal	
Transport of men	5	—
Ventilation	disregarded	
Total	55.5	45.5

thousand tons. The individual items of expenses listed in the table are taken arbitrarily.

A comparison of cost items yields a difference in favour of the second variant: $55.5 - 45.5 = 10$ thousand roubles, or 22 per cent.

Consequently, in this particular case the opening through a vertical shaft and crosscut appears to be more economical.

Since the difference in the mining cost and in the capital investment time is small, the standard efficiency factor is disregarded in the above comparison.

Problems arising in determining the method of opening-up a series of beds are solved similarly to early development of a single seam.

When working out the time schedule for the extraction of the mineral from individual beds of the series care should be taken that the overlying seam is not undermined by the subjacent one.

In opening-up a bed series an independent haulage entry is usually driven, and often one of the roadways serves as a mother (or gathering) entry, i. e. as a main haulage opening (see Fig. 10).

It is advisable that in opening-up a bed series with an unstable roof and bottom, and of contiguous thick seams containing self-igniting coal the group (gathering) entries be driven in the country rock.

2-7. FEATURES SPECIFIC TO OPENING-UP OF DEEP-LYING SEAMS

Because of intensive mining the depth at which some coal deposits are worked in the USSR has reached 900-1,000 metres. In the near future coal seams in the Donets basin will be mined at a depth of 1,200-1,500 metres. The conditions of mining coal deposits at a great depth differ substantially from those existing in shallow mines. In the former case the rock pressure is greater, the evolution of methane and the temperature in underground workings are higher, the angles of rock displacement are different. The methane content at a depth of 1,500 metres becomes as high as 40-75 cu m per 1 ton of daily output (compared to 10-30 cu m at a depth of 300-600 metres). The expected rock temperature in the Donets basin at a depth of 1,000 m is + 38° C and at 1,500 m + 54°C.

The higher cost of early development work in deep-lying deposits points to the expediency of constructing big mines with large mine-fields and extended service life. The higher methane content and temperature of rocks make it necessary to increase the cross-section and number of early development openings. Apart from the central location of the twin main shafts in mining deep-lying beds, flanking ventilating shafts should be put down and provisions made for independent aeration of the individual areas in the mine-field.

As the depth of mining increases so does the size of protective pillars left under surface structures and near the shafts, especially because of the possible flattening out of the angles of rock displacement zones. In some instances it may be expedient to locate shafts and surface structures outside the rock displacement zone. With large volumes of air used for dilution of methane and reducing the temperature at a great depth one should drive several openings in one level, some of them being utilized exclusively as airways for air currents moving at high velocities (up to 20 m/sec).

Opening-up of super-deep beds (over 1,500 m) should be correlated with the potentialities of single-lift hoisting down to a depth of 2-2.2 km (with the use of multi-rope hoisting plants). With a depth in excess of 2-2.2 km multi-stage shafts should be provided for early

development. When the rock temperature in deep-lying levels reaches a high mark one should envisage artificial cooling of the air through stationary refrigeration plants located on the surface and small mobile cooling units in individual levels, the latter permitting the air temperature at the faces to be reduced by 10-12°C.

2-8. CHOOSING THE SITE FOR SHAFT SINKING

This is a major problem in early development. The shaft location influences not only the scope of the work and the associated capital outlays, but has a direct bearing on the subsequent running expenses and on the date of putting the deposit into operation and the conditions of mining.

Location of the Main Shaft

The location of the main shaft depends, above all, on the adopted method of opening and the mining conditions, chiefly those of underground transportation and also on the local topography, which influences the disposition of railroad spur-lines and surface structures and plants. The choice of the shaft location may also depend upon the nature of the rocks through which it is to be sunk.

Let us consider the basic requirement that the mineral is hauled over minimum possible distances. The optimal location of the shaft as regards the least possible demands on the transport facilities usually also meets the requirement of minimizing expenses for mine pumping, ventilation, maintenance of workings and movement of men.

As stated above, along the strike the main production shaft should be put down on the line dividing the mine-field into two wings or flanks of equal size. This, however, holds true only for beds of regular thickness. In the event of a nonuniform distribution of the mineral along the strike one has to follow the rule for locating shafts proposed by Academician L. Shevyakov.

L. Shevyakov has proved that a shaft meeting the requirement of minimum possible demands on the transport facilities for haulage along the entries should be sunk from a spot facing a concentrated load Q_n, which, being added to all the other loads lying to the left of it, will yield a sum greater than the sum of all the loads lying to the right of it, and being added to the loads on the right side, will yield a sum exceeding that of the loads on the left side. This rule may be written in the form of two inequalities. The load Q_n which the shaft should face must satisfy two inequalities:

$$\Sigma Q_{left} + Q_n > \Sigma Q_{right}$$
$$\Sigma Q_{right} + Q_n > \Sigma Q_{left}$$

Figure 20 features an arbitrary skeleton diagram (*a*) showing the disposition of loads which is used in siting a shaft on the strike in a single seam with nonuniform distribution of reserves within the mine-field. The site for a shaft across the strike sunk in opening-up a series of beds is determined in a similar manner.

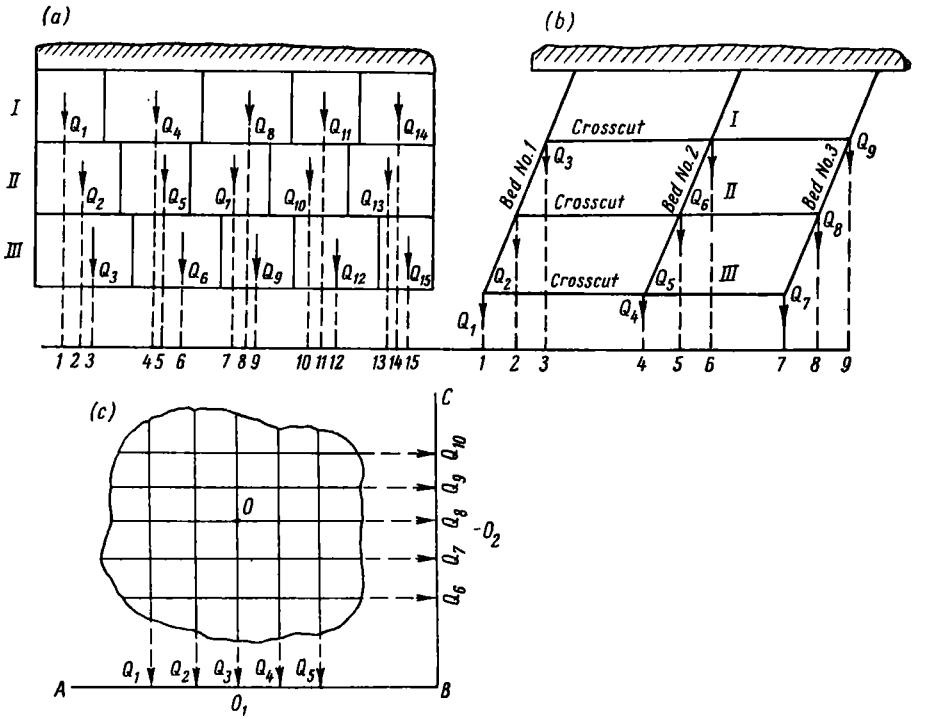


Fig. 20. Choosing the site for sinking main working shaft on the basis of requirement for minimal possible mine haulage:

a—distribution of concentrated loads along the strike (loads Q_1-Q_{15} correspond to mineral reserves in individual sections of the bed); *b*—distribution of concentrated loads across the strike (loads Q_1-Q_9 correspond to mineral reserves in individual levels of the bed); *I*, *II*, *III*—levels; *c*—distribution of concentrated loads Q_1-Q_5 in one direction and Q_6-Q_{10} in the other direction of travel

By way of example let us consider opening-up of three steep beds through a vertical shaft with crosscuts in each individual level. In this case (Fig. 20*b*) a horizontal line should be traced (parallel to the crosscuts) on which points of concentration of loads (intersection points of the crosscut with beds) in all crosscuts are then projected. Thereafter the site for sinking the shaft is established with reference to this arbitrarily chosen crosscut in the manner described above.

In siting the main shaft in a flat deposit one should proceed from the fact that tramming is ordinarily done in two mutually perpendicular directions. By plotting the points of load concentration on the lines AB and BC, parallel to the lines indicating the direction of tramming (Fig. 20*c*), and by following the above-cited rule the corresponding points O_1 and O_2 can be found. The position of the shaft in point O will be at the intersection of perpendiculars drawn from the points O_1 and O_2.

When selecting shaft location, the effect of surface transport, as well as that of underground transport, should be taken into account. If the spur-lines and sidings used for hauling out the mineral approach the shaft parallel to the strike of the deposit, or at a small angle to it, an additional consideration should be the concentrated load at the site of unloading (on the surface), which equals ΣQk, where ΣQ is the sum of the loads hauled underground, and k is a coefficient reflecting the reduction of expenses with the use of surface transportation as contrasted to underground tramming, which is set at 0.4-0.6.

Below we give basic information on other factors to be considered in selecting the site for a main production shaft.

Geological and hydrogeological conditions for shaft sinking. One should avoid sinking a shaft in quicksand, in zones of flooded karsts, in strongly shattered or very hard rock, for in such conditions the expenses incurred in putting down the shaft increase considerably, whereas the speed of sinking drops abruptly.

A preliminary exploration borehole should be drilled to ascertain the existing geological and hydrogeologicel conditions for shaft sinking at the proposed site.

Convenient and practical location of the bottom station and of other workings lying near the shaft with reference to its projected site. By convenient and practical location is meant the siting of underground mine workings near the shaft in stable rocks and the possibility of driving haulage entries with the necessary downgrade in the direction of the load traffic and run-off of the mine water.

Surface area suitable for building and subsequent operation of structures and plants. An area on the ground surface meeting the following basic conditions should be considered as suitable: it must lie outside the boundaries of the zones where shear and shift may occur when the deposit is mined to a maximum depth, or it should have a size and shape that would enable leaving a minimum of protective pillars of the useful mineral; the selected area should ensure the run-off of snow, storm, and mine waters, its ground should withstand the required pressure; it should be possible to build a railroad side line with the necessary upward and downward gradients, an aerial tramway, or any other road for out-of-mine transportation; a free (preferably low-lying) lot of ground for waste-disposal dumps should

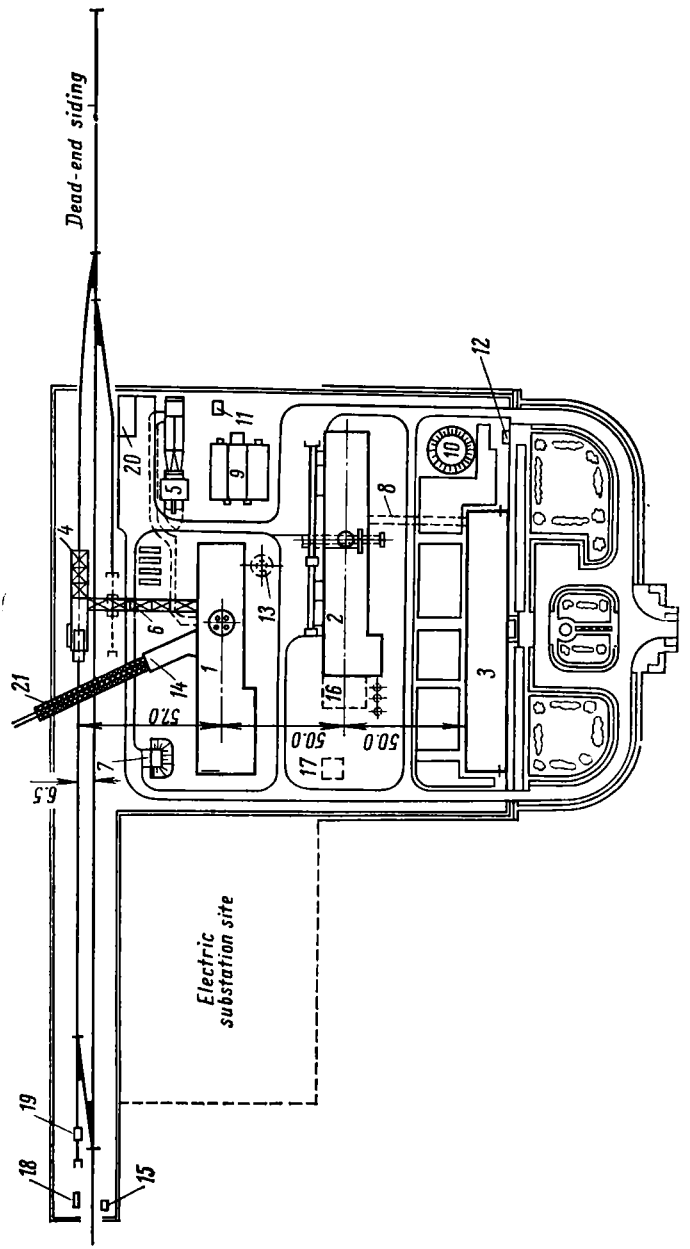


Fig. 21. General plan of a standard mine surface layout:

1—main shaft block; *2*—service shaft block; *3*—office-change and auxiliary services house block; *4*—receiving bins with sorting plant and loading points; *5*—fan house; *6*—gallery leading from main shaft block to receiving bins; *7*—storehouse for lubricating and lighting materials; *8*—underground passageway; *9*—mine water settling reservoir; *10*—fire-fighting water storage reservoir; *11*—chlorination room; *12*—entrance-gate booth; *13*—stack funnel; *14*—loading sites of the aerial tramway; *15*—switchman's cabin; *16*—compressor plant site; *17*—water-cooling tower site; *18*—trip-spotting hoist; *19*—trip-spotting carriage; *20*—unloading ramp; *21*—safety bridges of the aerial tramway

be available near this area, also a site convenient for building a workers' settlement, or for constructing a road to connect this area with an already existing or planned workers' settlement.

A level land plot or one with a unilateral slope of up to 5-6° with more or less smooth relief may be considered as suitable for building mine structures and plants.

In selecting a ground lot for mine structures use is made of plans drawn to a scale of 1 : 1,000 to 1 : 500 with the relief contour-lines,

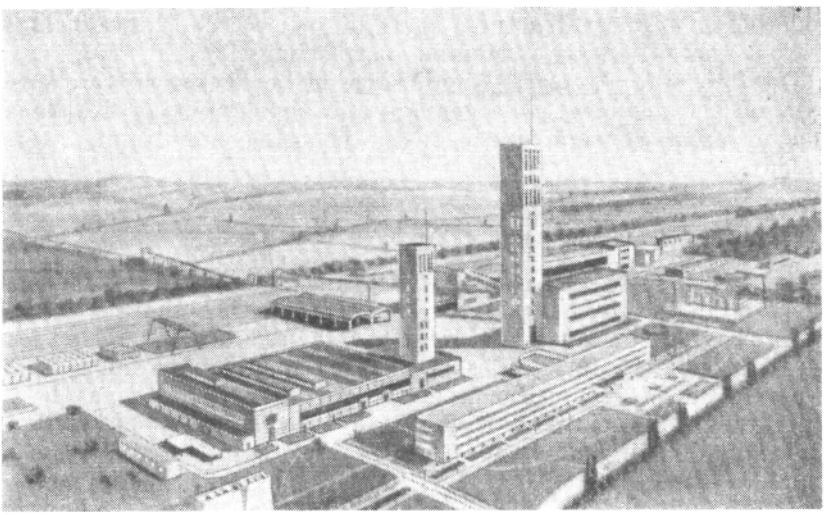


Fig. 22. Surface structures and plants of the new mine "Krasnoarmeiskaya-Kapitalnaya" in the Donets basin with production capacity of 12,000 tons of coal per day

size grid, wind rose, rivers and other bodies of water plotted thereon. The plans must indicate the boundaries of the deposit, its outcrops, the boundaries of rock subsidence and shift zones for a maximum depth of mining, and all the existing buildings and other structures.

To select a suitable land plot it is necessary to estimate its size tentatively, taking into account the area required for all major buildings and other structures, including standard- and narrow-gauge r. r. lines, highway, waste dumps and stock piles. In doing this use is made of data obtained from similar, already existing mines or planned mine layouts, and from rough sketches.

In practice the size of such land plots is adopted within the range from 10 to 100 thousand sq m, and in some instances even more, depending upon the location and production capacity of the mine.

A rough sketch showing the layout of structures and plants on the surface of a coal mine is given in Figs. 21 and 22.

Abandonment of protective pillars near shafts and other important structures. When choosing the site for sinking a shaft one must carefully compare the expediency of locating the shaft within the boundaries of the mine-field to be exploited, or beyond the zones of rock subsidence and displacement. The shaft may be put down within the limits of a large mine-field if the amount of the mineral left in the protective pillar is small. In leaving protective shaft pillars underneath surface structures it is of prime importance that the size of the pillars be estimated correctly.

The foregoing shows the significance of possessing comprehensive data on the geology, hydrogeology and topography of the deposit and of areas contiguous to it in choosing a suitable site for a shaft.

Location of Ventilating Shaft

In selecting the site for a ventilating shaft its location should be correlated with that of the main production shaft.

To provide for adequate ventilation and an emergency exit (as required by Safety Rules) each mine should have not less than two exits to the ground surface. The downcast fresh air may be passed through the main shaft equipped with a cage hoist, while the contaminated air will escape through the upcast ventilating shaft, which is often utilized for other purposes as well. When the main shaft is provided with a skip hoist the fresh air cannot be delivered through it, since during unloading of skips a large amount of coal dust contaminates the atmosphere. Here, fresh air should be supplied via a special shaft, or through the one designed for lowering and hoisting of men. The mutual arrangement of shafts may differ from one case to another. Three principal patterns of shaft disposition are used (see Fig. 23): (*1*) central disposition of the hoisting and ventilating shafts, which are closely spaced (but not closer than 30 m) and located in the centre of the mine-field; (*2*) central disposition of the hoisting shaft, with the ventilating shaft put down near the upper boundary of the mine-field; (*3*) diagonal disposition, when the hoisting shaft is located centrally, while one or, more often, two ventilating shafts are disposed diagonally, near the boundaries of the mine-field.

The main advantages of the central disposition of shafts are: the compact arrangement of surface structures; the possibility of quick drivage of a through-cut for aeration; the possibility of utilizing one of the shafts (usually the hoisting one) for raising another one; leaving a common protective pillar of the useful mineral.

A drawback of central disposition is the long distance the main air currents have to cover when upper levels are mined and also the considerable depth of the ventilating shaft.

The advantage of siting the ventilating shaft near the upper boundary of the mine-field is the smaller depth of the shaft, but in this case the merits of central disposition pointed out above are sacrificed.

The advantages of the diagonal disposition of the shaft (with two ventilating shafts) include: uniform operation of the mine fans,

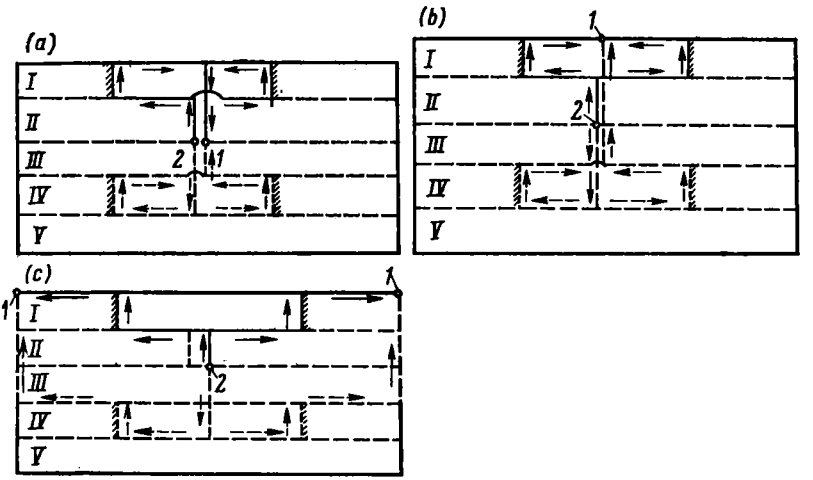


Fig. 23. Relative disposition of shafts in mining a gently sloping bed:
a—central disposition of shafts; *b*—ventilating shaft near the upper boundary of the mine-field; *c*—diagonal disposition of shaft; *1*—ventilating shaft; *2*—hoisting shaft; *I, II, III, IV, V*—mine-field levels. Solid arrow indicates direction of air flow in working upper levels, dashed arrow—in mining lower levels

since the distance the air current has to cover does not change substantially; feasibility of temporary aeration of the mine by means of a single ventilation plant (in case the second one is out of service).

The diagonal disposition of shafts has serious shortcomings: considerable outlays for the sinking of two ventilating shafts and for the installation and operation of two ventilation plants; the need for driving a long through-cut between the shafts prior to mining the mineral.

The above-listed shortcomings are quite important and for this reason the central disposition of shafts is used more often in working bedded deposits.

The diagonal location of shafts is favoured by: the presence of a worked-out area (above the mine-field), whose ventilation openings can

be employed in implementing the scheme of diagonal ventilation; utilization of effective methods of drilling in sinking shafts, which enables the required number of shafts to be driven on the boundary of the mine-field (along the dip) at a low cost and in a short time. In this case the distance travelled by the air currents can be considerably reduced, this being of great importance in mining gassy seams.

In working the first level of the mine-field lying close to the ground surface aeration is often effected through a series of vertical or inclined openings driven from the ventilating entry to the ground surface.

When selecting the site for an air shaft due account should be taken of the local topography, the nature of rocks, feasibility of conveniently locating shaft station workings, and the size of the required land plot, as in choosing the site for the main shaft. In many cases ventilating shafts are also utilized for other purposes, such as lowering supplies, hoisting country rocks, etc.

2-9. EARLY DEVELOPMENT OPENINGS IN BEDDED DEPOSITS

The principal openings excavated during early development include shafts, shaft or bottom stations and other shaft station workings, crosscuts, adits, permanent gravity (brake) inclines and slopes.

Shape, Cross-Sectional Area of and Basic Data on Driving Early Development Openings

The correct choice of the shape, cross-sectional area, the type and design of mine support to be used in early development openings is of prime importance, since it affects the cost of early development, the time needed for opening-up the deposit or part of it and the convenience of its subsequent exploitation.

The chosen shape of the opening's section should ensure cheap and easy drivage of workings in given conditions, it should also be in line with the adopted type of support, which is contingent upon the hardness and stability of the rocks and upon the planned service life of the opening.

The size of early development openings depends upon the minimal cross-sectional area needed for continuous movement of the mineral and country rock (muck), for safe and convenient passage of men, delivery of supplies and equipment and for the flow of the requisite volume of air.

In determining the size of the shaft section due consideration should be given to the cross-section of the shaft vehicles to be used (cages or skips). The shaft section dimensions are determined graphi-

cally, taking into account the size of the shaft vehicles and safety requirements as to the clearances between shaft vehicles, shaft reinforcement and support members, and also the size of the ladderway, pipe and other compartments of the shaft. Furthermore, the cross-sectional area of the shaft is checked for conditions of aeration and brought in line with the existing standards for shaft sections. The shaft section is basically given a rectangular shape when wooden timbering is used, and circular for stone, brick, concrete, reinforced concrete or metal support. Lately, vertical shafts are essentially made of circular shape.

Wooden timbering is used in shafts with a service life of up to 15-20 years and limited rock pressure. Metal, concrete or reinforced concrete support is employed in shafts with a long service life and considerable rock pressure. The cross-sectional area of a circular shaft is on the average 30% larger than that of a rectangular shaft of equal production capacity. In selecting the support and the cross-sectional area of a shaft its service life and repair costs should be taken into account.

When choosing the type of support consideration should also be given to fire safety and optimal ventilation conditions.

The number of shaft compartments ranges from 2 to 5 and seldom exceeds the latter figure.

According to the USSR standards for shaft sections shafts should accommodate two cages with an inner diameter of 5-6 m, two skips with an inner diameter of 4.5-5 m, four skips for 4, 6, 8 tons each with an inner diameter of 6 m, four or two skips for 4, 6, 8 tons each, and a cage (with a counterweight) with an inner diameter of 6 m, two cages accommodating a mine car of 3 tons capacity each, and a service cage hoist with an inner diameter of 8 m.

More often than not the size of the useful section of rectangular shafts is 10-20 sq m, and that of circular ones 16-50 sq m (the inner diameter being from 4.5 up to 8 m).

The thickness of wooden timber is 200-300 mm, that of concrete shaft support 300-400 mm.

The cost of shaft sinking is laid down according to standard rates contained in the Manual of Standard Estimate Rates. When making calculations the average monthly shaft sinking rate including timbering and reinforcement may be put down at 40-60 m. If gangue is mucked by several mechanical loaders at the face of the stope with simultaneous performance of basic mining operations the average shaft sinking rate can be raised to 80-100 m per month (the maximum shaft sinking rate attained in the USSR at the time of writing was as high as 390 m per month).

In very stable rocks adits and crosscuts are driven without timbering, the roof being given a domed shape. In medium-stable rocks

and with a short service life these openings are mostly trapezoidal in cross-section and supported by wooden timber frames made of props 200-300 mm in diameter. In unstable rocks and with a service life in excess of 10 years metal, stone or concrete support is employed. In recent years prefabricated reinforced concrete support has been widely used. With stone and concrete support the cross-section of an opening is made of rectangular shape with a dome-like roof, the thickness of the concrete usually being 150-300 mm. With support of prefabricated reinforced concrete members the section of the opening is rectangular, trapezoidal, or polygonal. The arched shape is common when yielding metal support is used. In individual cases these types of support are also employed with relatively short-life workings (3-5 years), when, because of the high rock pressure, wooden timbering is unacceptable. Long-life horizontal workings are supported by metal frames, concrete, or prefabricated reinforced concrete members.

When deciding on the sectional size of adits and crosscuts account should be taken of the overall dimensions of mine cars and locomotives, the needed number of haulage tracks, the width of the passage for men, the clearance between mine cars or locomotives and timbering, and the volume of air required to aerate underground workings during exploitation.

The minimal useful section of adits and crosscuts should be 4.5 sq m with wooden or steel support, and 4 sq m with stone or concrete support.

The commonly adopted size of the useful section in adits and crosscuts is 4.5-5 sq m for single-track, and 7.4-9.6 sq m for double-track openings.

Entries made in coal have approximately the same sectional area.

In the USSR tunnelling machines are widely employed in making entries and drifts.

The most widely used type of tunnelling machine in the USSR is Model ПК-3 (ПК-3м). In 1965 of the 524 tunnelling machines in operation 370 were of the ПК-3 (ПК-3м) model (Fig. 24). In that year tunnelling machines had driven 510 km of mine openings, the average rate of advance being 179 m per month.

The maximal rate of advance in horizontal workings in coal, achieved in the USSR with the aid of tunnelling machines was as high as 2,523 m per month. In calculations the average monthly rates of advance in crosscuts and adits should be put down at 100 m, and for strike entries in coal 150-200 m.

Expenses to be incurred in driving adits, crosscuts and other horizontal mine workings are estimated in accord with the specified rates.

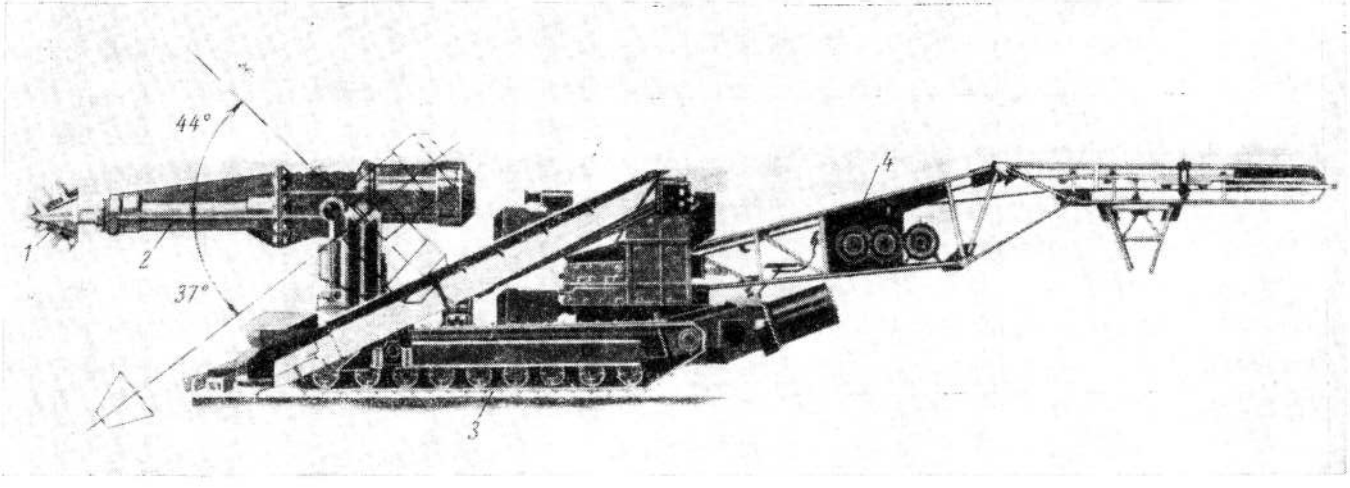


Fig. 24. ПК-3м type drifting machine:
1—operating mechanism; *2*—boom; *3*—caterpillar drive; *4*—reloader

Table 2

Tunnelling Machines Used in the USSR

Type	Application		Method of extraction	Sectional area of opening, sq m	Weight, tons	Maximal capacity, m/hr	Electric power, kW
	Rock hardness (M. Protodyakonov's scale)	Maximal angle of opening's slope, deg					
ПК-2м (ПК-2с)	$f \leqslant 2$	±15	Breast all-round	5.6-7.92	10.34	6	40.1
ПКГ-3	$f \leqslant 2$	+30-15	Ditto	4.3	13	8	69.5
ПК-3 (ПК-3м)	$f \leqslant 4$	±15	Selective	5.3-12	10	4	66
Karaganda 7/15	$f \leqslant 4$	±15	Breast all-round	7-14.5	36.8	14	176
ПК-7	$f \leqslant 4$	±15	Selective	4-8.2	10.65	4	64
ПК-9	$f \leqslant 4$	±15	Ditto	6-16	28.6	16	192
ШБМ-2	$f \leqslant 4$	±15	Breast all-round	8-10	40	3	106.5

Arrangement of Shaft Stations and Their Characteristics

A shaft station is an aggregate of mine workings connecting the shaft with main haulage and air openings, it serves as the principal terminus for all loads delivered to the ground surface and for supplies transported from the shaft to work-places. In the vicinity of the shaft station there are all major underground service rooms, such as the pump room, the underground sheds for electric mine locomotives, the electric substation, the rest room, etc.

The shaft or bottom station consists of the following basic sections: the loaded-train branch of the station which includes openings where loaded mine cars delivered from work-places are stationed pending their unloading in car-tippers or hoisting in cages to the ground surface; the empties branch of the station consisting of openings into which empty mine cars are trammed after being tipped or pulled out of the cage; by-passes connecting the load side with the empties side of the station or with other haulageways and intended for train traffic or for the switching of electric locomotives.

If two hoist compartments are available on the load side of the station two main openings are distinguished: the skip and the cage branches.

Table 3

Minimal Rates of Advance in Drivage of Mine Workings According to 1963 Standards Approved by the USSR State Committee for Construction

Description	Units of measurement	Minimal advance rates with the use of loading machines	
		In coal and combustible schist mines	In metalliferous mines
1. Vertical shafts	m/month	35	35
Inclined shafts	ditto	50	50
2. Entries made in coal . .	ditto	100	—
Entries made in coal with undermining of rock . . .	ditto	80	—
3. Crosscuts and stone drifts	ditto	60	80
4. Gravity inclines	ditto	90	—
5. Slopes	ditto	70	—
6. Permanent ore-chutes and raises	ditto	—	40
7. Junctions of mine openings, and also junctions of shafts with bottom stations (per 1 junction)	cu m/month for effective section	250	300
8. Shaft stations and rooms adjacent to them	cu m/month for effective section	250 (per one heading face)	1,000
9. Service rooms outside shaft stations	ditto	—	400
10. Shaft reinforcement . . .	m/month	200	200

Note: 1. The standard rates may be reduced:
(a) in shaft sinking — by 40%;
(b) with bleeding fire-damp in workings driven uphill, the water inflow exceeding 200 cu m per hour, or in workings driven downhill with water inflow of more than 100 cu m per hour — by 20%;
(c) in metalliferous mines when driving horizontal workings and excavating rooms and chambers which require timbering — by 25%.
2. In coal mines standard rates of advance in openings which require no timbering should be raised by 30%.
3. When two or more factors reduce the advance rate of mine openings, only one, the largest, should be used.
4. With the use of tunnelling machines for driving strike entries in coal standard rates of at least 200 m per month should be adopted.

In mines of low production capacity tracks for loaded and empties trains are laid in one mine opening, with no by-passes.

The shaft station is arranged normally or parallel to the main haulageway. Depending upon the nature of the mine-car traffic the shaft stations are provided either with stub or circular track lines. Bottom stations may be of the cage- or combined skip-and-cage type.

The basic layouts of shaft stations are shown in Fig. 25. According to the pattern of movement of loaded and empties trains, stations

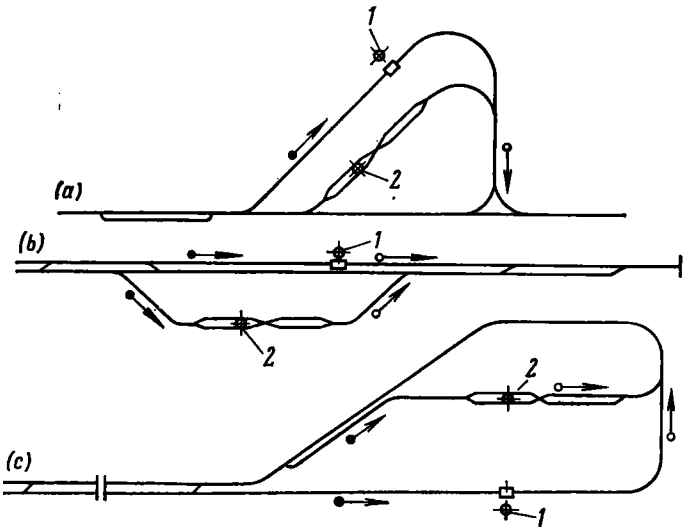


Fig. 25. Standard shaft station layouts:
a—with circular mine traffic; *b*—with shuttle traffic; *c*—loop-like layout of shaft station; *1*—main shaft; *2*—service shaft

are classified as those with circular traffic, with shuttle movement of trains, and with stub-tracks. Shaft stations with one-way circular traffic are said to have a loop-like arrangement. In shuttle-type bottom stations loaded trains are brought to the shaft from both sides along an opening running parallel to the main haulageway. Shaft stations with stub-tracks have a layout similar to the shuttle-type ones, the difference being that loaded trains move to the shaft from one side only.

As compared to the bottom stations with circular movement of trains the stub-type ones have a number of technical advantages. The rectilinear arrangement of the stub-type station favours normal performance of operations there, particularly on gravity tracks. The drawback is the necessity of more complex switching of mine

cars, but with complete mechanization of haulage at the shaft station this disadvantage is of no great significance.

In the majority of modern coal mines of the Soviet Union mine cars are moved mechanically within the limits of the shaft station.

When estimating the overall extent of shaft stations the length of the loaded-train and empties track branches of the station is designed to accommodate 1.5-2 trains on each side. Stations with cage-hoisting arrangement at service shafts are meant to hold a smaller number of mine cars. The size of shaft stations (less service rooms) depends upon the type of the station and the production capacity of the mine.

In each individual case the size of a shaft station in a coal mine can be calculated graphically by plotting the workings and estimating their size.

Roughly, the aggregate volume of openings and service rooms at a shaft station can be determined by using the following formula:

$$V = 2\sqrt[3]{A^2} + 0.01A, \text{ cu m}$$

where A is the annual output of the mine in tons.

Workings at the shaft stations are more often than not provided with concrete or prefabricated reinforced-concrete support. According to the Safety Rules support at the site where the station joins the shaft should be made of fireproof material.

Service Rooms at the Shaft Station

Service rooms and ancillary workings at the shaft station include: a bunker arrangement (pocket), pump room, power substation room, electric locomotive shed, underground repair shop, storage room for fire-fighting materials and appliances, rest room, underground storage room, underground control room, first-aid room, battery charging room, tipper-and-pusher rooms, drain sumps.

In a large-output mine the layout of service rooms is as shown in Fig. 26. In low-output mines the number of rooms is cut down considerably, their layout is simplified; they are situated in the immediate vicinity of the shaft. Below we briefly describe the arrangement of principal service rooms.

Shaft pocket. With skip hoisting of the mineral each extracting level is provided with a shaft pocket designed to hold a certain tonnage of the mineral. These pockets are either low-capacity (for the payload of one skip), or large-capacity (for one train load of the mineral, sufficient to keep the hoisting plant busy for 1-1.5 hours).

A low-capacity shaft pocket is used when the service life of a production level is short, the country rocks insufficiently stable and breaking up of the mined mineral is undesirable (Fig. 27).

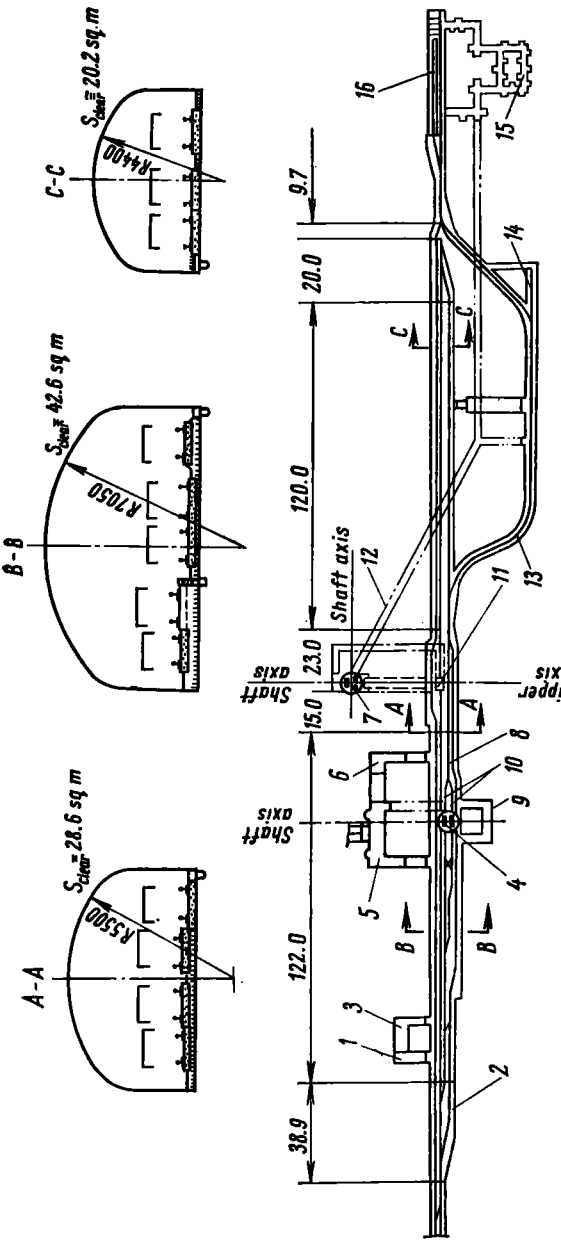


Fig. 26. Switchback shaft station layout of new design with one rectilinear multitrack arrangement:
1—control room; *2*—parking site for man-riding cars; *3*—first-aid room; *4*—service shaft, $d = 6$ m; *5*—pump room; *6*—power substation; *7*—main shaft, $d = 5.5$ m; *8*—measuring stop; *9*—rest room; *10*—catches; *11*—tipper-and-pusher room; *12*—ventilating through cut; *13*—locomotive shed and battery charging room; *14*—repair room; *15*—powder room; *16*—fire-fighting train shed

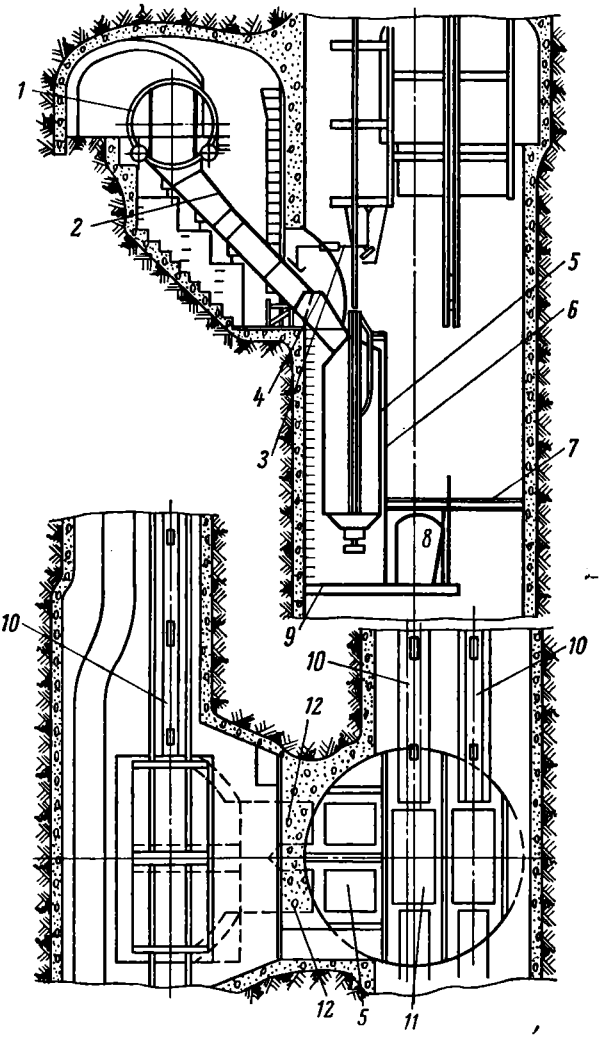


Fig. 27. Low-capacity bunker arrangement for skip hoisting:

1—tipper; *2*—coal pocket; *3*—safety shutter; *4*—feed spout; *5*—skip; *6*—lining; *7*—safety platform; *8*—sump drainage pump room; *9*—protective stage to prevent access of coal into sump; *10*—car pusher; *11*—cage; *12*—loading arms

A large-capacity pocket is built in a long-life production level, in stable (rigid) rocks and in mining a mineral which is not subject to compaction (packing) and therefore can be reduced to small fractions.

Pump room and storage sumps. The pump room is located in the immediate vicinity of the shaft so as to cut down the length of the pipeline from the pump room to the shaft.

Storage sumps are directly adjacent to the pump room and often have a substantial length. Pump rooms accommodate the principal mine drainage plants. Storage sumps are reservoirs receiving the inflow of mine water; they are usually designed to hold water accumulated within six consecutive hours. Storage sumps consist of two compartments, one of which can be shut off for cleaning. They are reinforced with frames or concrete support. The floor elevation of the sumps should be around 5.5 metres below that of the pump room and 5 metres below the elevation mark of adjacent haulageways. The floor of the pump room is usually raised 0.5 metre above the floor level of the shaft station.

Storage sumps are cleaned through an inclined manway (passageway) along which mine water flows into them. The pump room should accommodate not less than three pumping units, each with a rated capacity sufficient to handle daily inflow of mine water in twenty hours. Each pumping unit is provided with an independent drain pit sunk to a depth 1-1.5 metres below the floor level of the sump. By means of a collector all suction pits are connected with a receiving pit, which in turn opens into the main water sump. The purpose of the receiving pit is to distribute among the drain or suction pits the water flowing from the storage sump. Opening and closing of the storage sump and suction pits is done by manipulating slide valves mounted in the receiving pit. The valves are controlled from the pump room.

An inclined 1.5×1.5 m manway is run from the pump room to the shaft on the side of the ladderway to accommodate the discharge pipes. It also provides free passage to the pump room in case the shaft station is flooded. Two water-tight doors are set up in the pump room to seal it off from the shaft station in the event the latter is flooded. The support in the pump room should be fireproof.

The pump room arrangement is shown in Fig. 28.

Power substation. This accommodates transformers and converter units. The transformers step down the electric voltage. Three-phase alternating current is supplied underground from the surface through special cables at 3,000-6,000V and working units are fed at 380V. Motor-generator sets or mercury-arc rectifiers convert the alternating current into direct current to supply power to trolley electric locomotives or to charge storage batteries of battery locomotives.

The electric power substation is usually set up near the pump room, since pump motors are the main consumers of electric power.

Underground locomotive shed with repair shop. This is a site for parking, inspection and running repair of electric locomotives, and also for charging storage batteries where battery locomotives are used (Fig. 29).

The locomotive shed may be disposed in a special working or in an enlarged portion of some lateral opening in the shaft station. The shed should

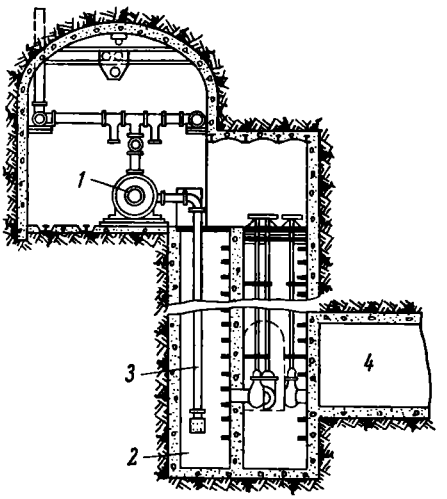


Fig. 28. Underground pump room:
1—pumps; *2*—water draw well; *3*—suction, or inflow end; *4*—part of sump contiguous to pump room

be provided with a separate exit and entrance. The repair shop of the shed has an inspection pit for electric locomotives and lifting

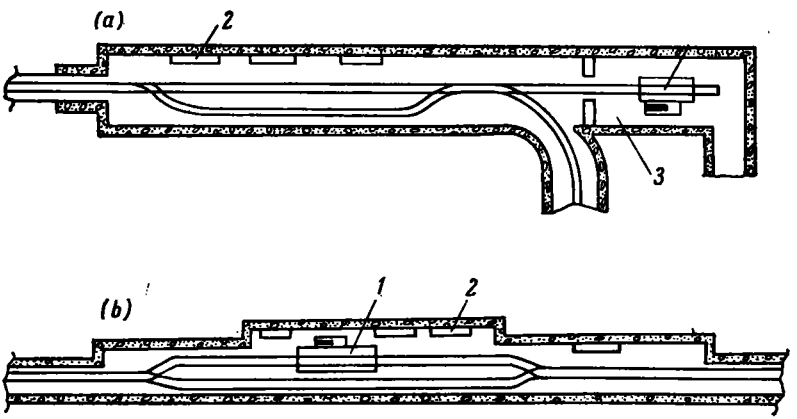


Fig. 29. Schematic layouts for electric locomotive sheds:
a—shed in a special working; *b*—shed in an enlarged portion of a mine opening; *1*—inspection pits; *2*—tool boxes; *3*—repair shop

equipment, such as a crane and pulley blocks. The support of the electric locomotive shed should be fireproof.

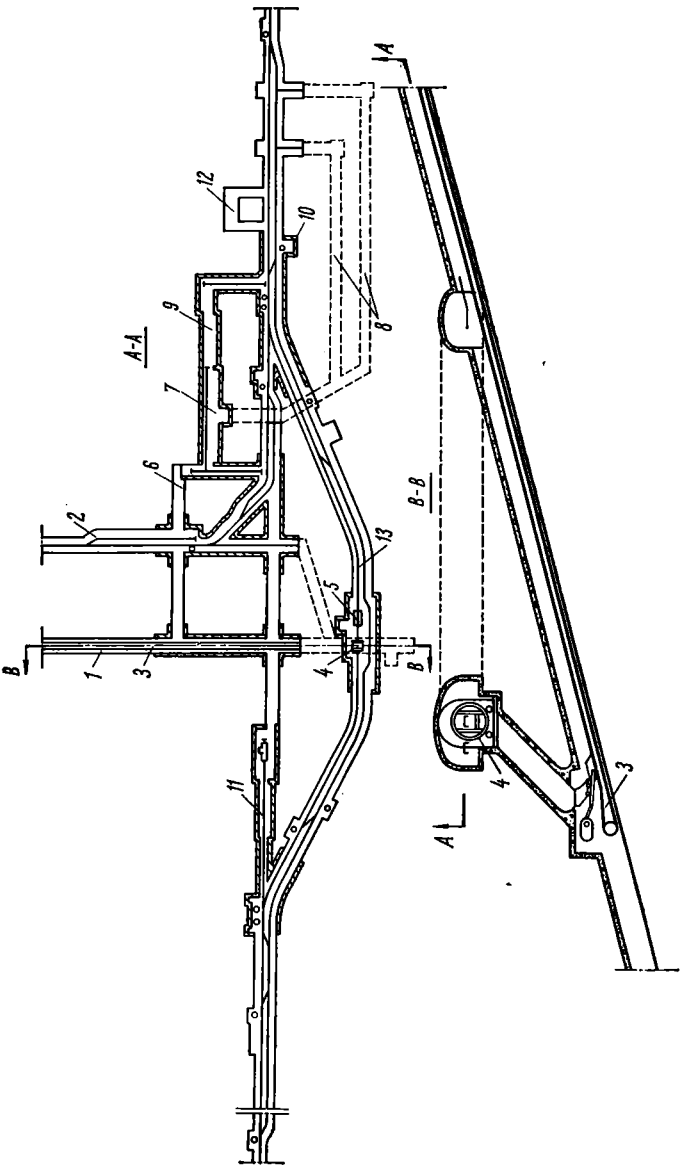


Fig. 30. Main station in inclined shaft:
1—main shaft; *2*—service shaft; *3*—conveyer line; *4*—tipper; *5*—car pusher; *6*—rest-and-waiting room; *7*—pump station; *8*—sump; *9*—power substation; *10*—central underground control room; *11*—underground garage; *12*—first-aid room; *13*—by-pass

Service rooms. Rest-and-waiting, central control, first-aid and other rooms serve ancillary purposes. They are set up in large-output mines, have a limited size and a simple layout. The underground powder room should lie at a distance not less than 100 m from the main shaft and not less than 25 m from mine openings used as permanent passages for men and must be aerated by a separate air current passed along a special airway to the ventilation level.

The aggregate size of all service rooms excavated around the shaft station (shaft pockets, pump station, storage sumps, power substation, locomotive shed, etc.) comprises about 50% of the total volume of the shaft station.

In each individual case the size of workings at the shaft station can be defined more precisely by estimating the volume of mine openings in individual levels. The layout of bottom stations in inclined shafts is shown in Fig. 30 (the main shaft is equipped with a conveyer hoisting plant and the service shaft with a mine-car rope-haulage hoist).

The weighted mean cost of 1 cu m of mine workings and service rooms may be tentatively estimated thus:

	At the time of construction, roubles	In an active mine, roubles
Timber	48	38
Metal	54	43
Concrete	70	55
Prefabricated reinforced concrete	72	57

2-10. TOTAL OUTLAYS AND TIME-SCHEDULE FOR CONSTRUCTION OF A COAL MINE

In planning, the outlays for the construction of a mine are preliminarily determined in a target plan and ultimately fixed in a general estimate attached to the technical project. A rough estimate of the total expenditure for the construction of a mine may be made on the basis of itemized expenses for mining operations, whose volume and cost may be calculated in round figures for the particular project or determined from estimated or standard norms and rates. Below we set out an approximate structure of capital outlays for the construction of a coal mine with a production capacity of 600 thousand tons per year.

I. Industrial construction (without spur-lines)

	Expenditure, %
Preliminary work	1.5
Mining operations (drivage of early development openings for exploitation of entire deposit)	22.5
Construction work (building surface structures and plants)	17.0
Equipment and its assembly (hoisting, mine drainage, transport and other facilities)	19
Stocks	1-1.5
Sundry	3.5
Total for industrial construction	65.0
II. Housing and public utilities	35.0
Grand total	100.0

The cost of the total set of surface mine structures includes that of industrial structures and plants, equipment and housing.

The expenditure for the conduct of mining operations (excavation of early development openings) comprises approximately one-third of the aggregate sum allocated for industrial mine construction.

The time necessary for the construction of coal mines varies from 1 to 3 years, depending upon the mine output and the conditions of construction.

2-11. PLANNING EARLY DEVELOPMENT OF A BEDDED DEPOSIT AND PREPARATION OF AN INTEGRATED PROJECT

Mining of a deposit, including its opening-up, should be done in accordance with a previously drawn up and approved plan. When preparing a project for mining large sheet deposits an integrated plan for the development of the corresponding district should first be compiled.

According to the instructions issued to the coal-mining enterprises of the USSR an all-round program for each district should include all basic technical and economic data concerned with the development of the district, such as the size of the mine-fields, the production capacity of individual mines, the development of coal mining, coal preparation, railroad transport, the water and power supply of the district, the construction of central machine repair shops, the required labour force, the construction of residential houses, cultural and public service buildings and public utilities.

An integrated project for a district actually forms a basis for he planning of individual mines and associated enterprises necesary for the development of coal production in the district, for

drawing up a program of detailed geological and exploratory work and for the planning of mine construction. Integrated projects are also prepared for mining the deposits of other useful minerals. In addition to integrated projects for individual districts, all-round plans for entire basins are drawn up.

2-12. PREPARATION OF A MINE-FIELD FOR STOPING

By preparation of a mine-field for stoping is understood drivage of mine workings after opening-up of a bed or seam so as to be able to start stoping operations.

In the USSR two methods of preparing mine-fields are used in mining coal deposits, viz. the *level* and *panel* methods. The level method is employed most widely. Development by panelling is partially used in mining gently dipping seams (35-40% of all stoping faces in the Donets basin and 45-50% in the Kuznetsk basin) and is the only method applied in working flat seams (the Moscow basin, etc.).

Diagrams showing level and panel development of a mine-field were given in Figs. 1 and 2 on p. 12.

The basic data concerning level preparation (height of the level interval, sequence of mining operations, etc.) have been discussed above. Let us now briefly consider the panel method with special reference to slightly inclined seams.

Each panel, usually of rectangular shape, is serviced by transport and ventilating workings. The width of the panel (its extent along the strike) is mainly contingent upon the transport facilities used in horizontal workings and is taken to be up to 1,000 metres and more. The length of the panel (its extent on the dip) chiefly depends upon the transport facilities employed in inclined workings and is set down at 1,000 metres and more. In recent years the size of panels has been considerably increased because of the wide use of conveyer transport.

A panel is divided into stages (see Fig. 2). A stage is a portion of the panel extracted at a time. Panels are mined in a consecutive order: the first to be worked are panels lying on the rise of the mine-field, simultaneously in the right and left wings, and then those disposed along the dip of the mine-field. Working of the stages in the up-dip portion of the mine-field is mostly done in the ascending, and less frequently in the descending order (in very gassy mines). The stages in the down-dip portion of the mine-field are mined in the descending order.

Each stage, as well as each level, can be extracted by advance or retreat mining. The retreat mining of a panel (analogous to that of a level) offers great advantages. As the size of the mine-field

increases on the strike, the panel method of development becomes more economically efficient (the expenses for the upkeep of workings rise at a slower rate than in the case of level preparation). Panel development also creates favourable conditions for increasing the footage of working stopes and the utilization of powerful electric locomotives for the haulage of the mineral in one level.

A good illustration to this is the mining of a down-dip portion of the mine-field by the panel method with the use of conveyer inclines practised at the Oktyabrskaya mine in the Donets basin (see Fig. 31). The mine-field has been opened up through two central shafts. The skip shaft, $d = 5.5$ m, is equipped with two skips, and the cage shaft, $d = 5.5$ m, has a double-cage hoist. The central (main) panel in the down-dip portion of the mine-field is worked by 2-3 faces with a planned output of 1,000 tons of coal per day (the field lies below the 385 m level and extends 4,400 m along the strike and 3,500 m on the dip, its wings being 2,200-2,400 m long). The panel is divided into stages with an inclined height of 220 m (one face). The stage is worked by longwalls in the direction from the boundaries to the conveyer incline. Within the stage development is accomplished with the use of entry-driving machines of the ПК-9Р model with separate extraction of coal and rock. Stoping is done with the aid of mining machines КМ-87 in combination with a narrow-cut heading machine 4К-52. In inclined workings men are transported in single-rope haulage cars, and along horizontal workings, in cars hauled by electric locomotives. The Oktyabrskaya mine exemplifies concentration of mining operations in a large-sized panel with a long face and all-round mechanization.

Planned figures for the operation of the mine (after completion of the development work):

1. Average daily output	6,000 tons
2. Number of active working faces	3
3. Planned average daily production per 1 face	around 2,000 tons
4. Efficiency (output) per stoper	140 tons per month

The following general conclusion can be made concerning panel development with an easy dip: the panel method, especially with the use of all-round conveyer transport within the panel, provides conditions for a better organization of operations in working faces, higher efficiency of labour and increased production of the mineral. The final choice between the methods of panel and level development is made by comparing their technical and economic efficiency. There is reason to expect in the near future a wide-scale introduction of panel development in gently sloping beds.

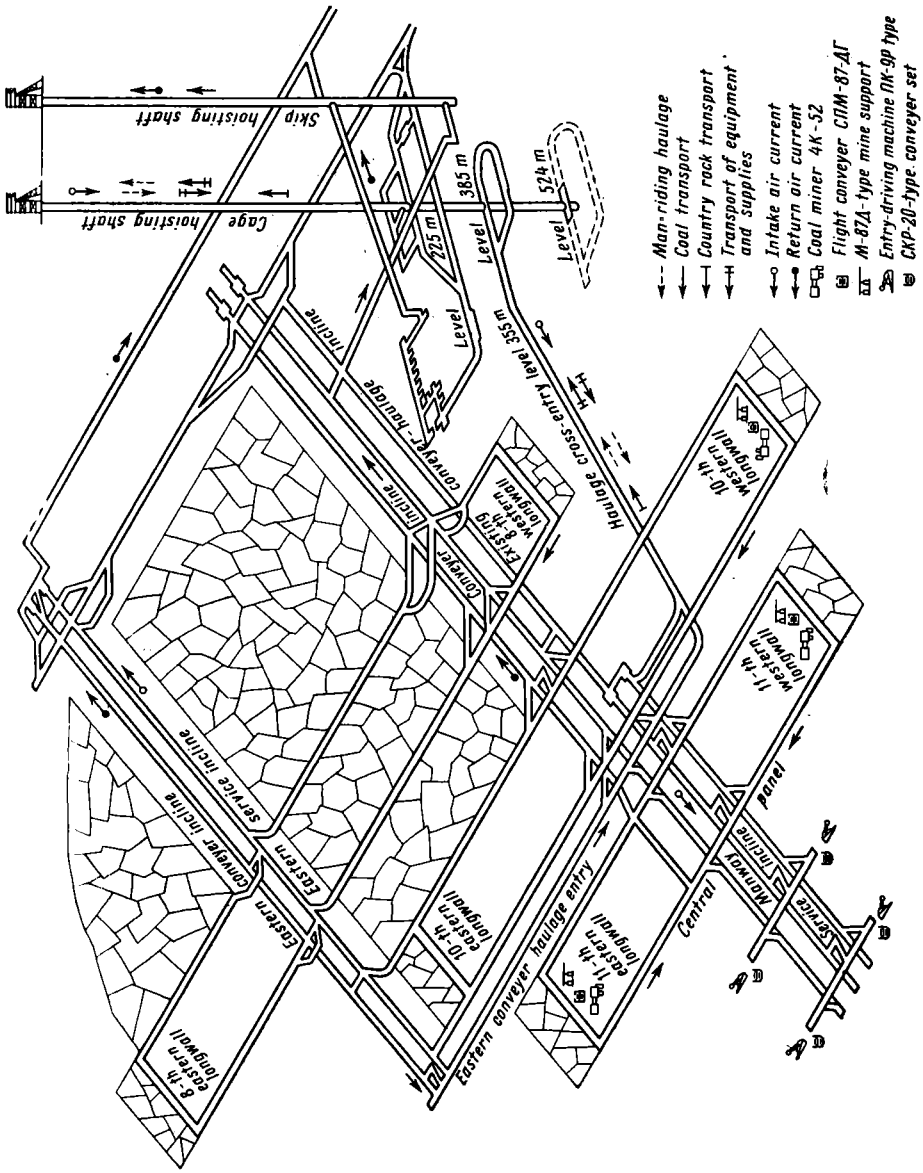


Fig. 31. Opening-up and panelling (blocking-out) at the Oktyabrskaya mine (Donets basin)

Panel Method of Development in Mining Horizontal Seams

As far as panel development of horizontal seams is concerned the Moscow coal basin is the most typical. A flat pitch of beds is also characteristic of a number of other deposits, such as the Cheremkhovo and Kansk coal deposits, the Baltic shale deposits, etc. With panel development of flat deposits two or three main entries servicing the entire mine-field are driven in the direction from the shaft to the mine-field boundaries (see Fig. 3). One of the entries is used for tramming and the others for ventilation. From the main entries haulage, ventilating and face entries, servicing the panel, are driven up to the mine-field boundaries. Butt entries delineating producing sections are pushed forward at right angles to face entries.

Panels may be made unilateral or bilateral (with respect to the face entry). Bilateral panels are more economical.

Based on the least cost, the size of a bilateral panel can be determined by using A. Sudoplatov's formula

$$X_0 = 2\sqrt{\frac{KS}{\frac{r}{L_b}+\frac{r}{L_a}+qSpC}}$$

where X_0 = the most advantageous width of panel, m;
K = cost of driving 1 m of two face entries (the tramming and ventilating ones), roubles;
S = width of the extraction area representing the summary length of the working face (or of two double working faces) and of butt entry stopes m;
r = cost of maintaining 1 m of butt entries for 1 year, roubles;
L_b = advance rate of butt entries per year, m;
L_a = advance rate of active stopes per year, m;
q = delivery cost per 1 t/m of coal along conveyer entry, roubles;
p = average yield per 1 sq m of seam, tons;
C = coefficient of recovery of mineral reserves in panel.

The width of a bilateral panel is set at 600-800 m. Mention has already been made of the direction of stoping and panel size with special reference to the mining of horizontal seams with a regular mode of occurrence. With an irregular mode of occurrence, especially with varying elevation marks of the bottom of the bed or of sheet-like deposits the size and directions of extraction of individual panels should be brought in line with the transport and mine drainage requirements, i.e. with the track and water run-off gradients.

2-13. FEATURES OF OPENING-UP OF PLACER DEPOSITS MINED BY THE UNDERGROUND METHOD

General Information on Placers

Placers are secondary deposits formed as a result of disintegration of primary occurrences.

Such deposits are loose or clay-cemented accumulations of rock and contain valuable components (gold, platinum, cassiterite, diamonds, etc.).

Placer deposits crop out or occur close to the ground surface. The underground method of mining is used in deposits occurring at a depth of 15-25 metres and more.

Placers are divided into two types—eluvial, or residual, formed at the site of their origin, and alluvial, or mass-movement placers,

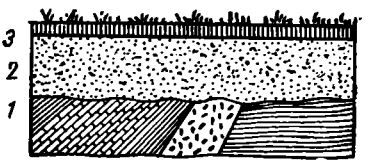


Fig. 32. Section across a placer deposit:
1—bedrock; *2*—auriferous sands; *3*—peat

transported (chiefly by water streams) over a considerable distance from the primary deposit. Alluvial deposits are more common.

Occurrences of alluvial placers are closely associated with river systems, being habitually confined to the bed of river valleys or other water bodies. Considering an alluvial placer as a sheet-like mineral deposit the following portions can be distinguished in the vertical direction (Fig. 32): bedrock, or rim rock of the placer, i.e. basement primary rocks of the placer, or products of their disintegration; a stratum, or sands—a layer of detrital deposits of the placer containing a commercially useful component; peats—sediments capping the bed and containing no useful components or containing them in amounts insufficient for commercial utilization.

Usually there is no clear-cut borderline between the bed and peat and it is established on the basis of assays and technical and economic calculations.

In the cross-section of a placer one can distinguish the so-called *stream* and *edges*. By the stream is meant the band of sands richest in valuable components. The farther away from the stream the smaller is the proportion of useful components in the sands, and the latter gradually becomes totally barren. The material making up a placer is ordinarily divided into two categories of sediments: sandy-pebbled (river gravel) and argillaceous.

Sands rich in a useful component are largely found near the bedrock. At times valuable components penetrate the bedrock through crevices, thus enriching its top portion.

Opening-Up Placer Deposits

In early development the sheet-like deposit of a placer is broken up into small mine-fields (Fig. 33). Depending upon the mining depth and the adopted transportation methods mine-fields are

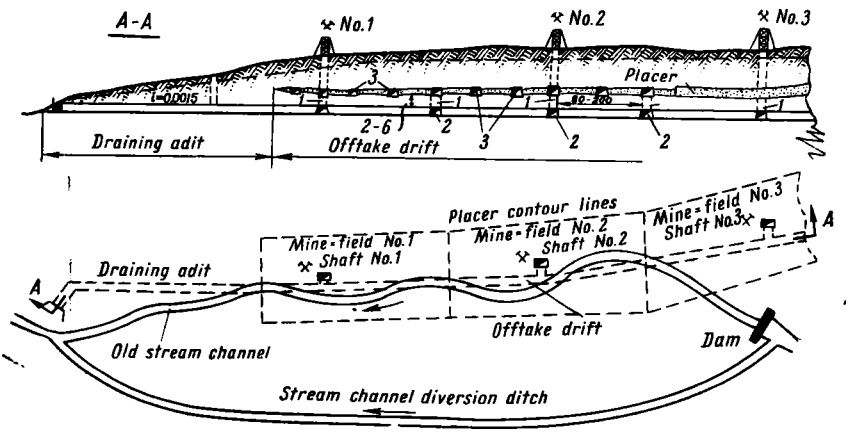


Fig. 33. Opening-up through individual vertical shafts and drainage of placer with free runoff:

1—vertical openings driven up from crosscuts; *2*—drifts made in the bedrock; *3*—drifts cut in the placer itself

made from 200 up to 800 m long (along the placer); larger lengths are used with placers exceeding 30 m in depth. The width of the mine-field is usually chosen to equal that of the placer and only in rare instances, when the breadth of the placer is very great, it is cut into two or three mine-fields crosswise.

The final size of the mine-field along the strike is based on technical and economic estimates with due allowance for initial outlays and running expenses, including expenditures for installation and upkeep of washer units.

Formerly, mine-fields of placer deposits were usually opened up through vertical shafts. Since drivage of a drift starting from the shaft down the ancient river bed entails great difficulties mine-fields are customarily made unilateral. If the payable portion of a placer has a considerable length and it is divided into several mine-fields for mining purposes, the development of the underlimb

of the mine-field may be conducted through a drift pushed from the subjacent shaft of a neighbouring mine.

In recent years placers have usually been opened up through inclined shafts, the mined sands being brought directly to the surface by conveyer lines. When the local relief is favourable, a placer can be opened through an adit, which is usually driven along the valley thalweg from the lower boundary of the placer. At the outlet to the ground surface the adit is excavated by the open-cut method as a trench with top covering. Mine-fields opened through an adit are made unilateral.

Prior to mining a placer deposit it is important to divert surface water from the mining area and drain the placer (Fig. 33). To this end the river is ordinarily diverted by building a special dam. To prevent infiltration of water the dam foundation is made in a compact, water-tight ground. The outfall ditch is made at a distance of 100-200 metres or more from the minable portion of the placer, along one of the valley banks. Sometimes water run-off troughs, or races are employed for the purpose.

Ground water is diverted to the surface along drainage mine workings by a free run-off, or with the aid of pumps. The final choice of the placer mine-field opening pattern is based on calculations by using the method of comparing variants.

A promising method of placer opening in underground mining is through an inclined shaft with conveyer delivery of sands.

CHAPTER III

OPENING-UP AND DEVELOPMENT OF ORE DEPOSITS

3-1. PECULIARITIES OF OPENING-UP ORE DEPOSITS

Most of the definitions and rules that concern opening-up coal deposits may also be applied in solving problems associated with the opening-up of ore occurrences, though the conditions of their mining have some special features. Below we give additional information pertaining to the opening-up of ore deposits, which, in contrast to sheet deposits, have a limited size and exhibit more frequent and marked changes in their mode of occurrence.

Determination of Danger Zone Boundaries and Construction of Safety Pillars in Mining Ore Occurrences

Let us consider some brief information on the delimitation of danger zones and construction of safety, or protective pillars. More detailed data on the subject are set forth in special surveying manuals.

Delimitation of a Danger Zone

First of all we determine the bounds of a zone dangerous with respect to the disposition of surface structures (Fig. 34). The danger area on the surface, in turn, usually includes two zones, the caving zone (central portion) and the ground movement zone (peripheral portion). In fixing the boundaries of the caving zone one is guided by fractures and cracks on the ground surface. The ground movement zone extends from the caving zone to the intact solid mass of rocks in situ. The ground movement zone is characterized by a more uniform dislocation (subsidence) of rocks and the absence of discontinuities in the ground surface. The boundaries of the caving and ground movement zones are plotted by drawing, at definite angles, lines from the lower border of the production stopes. The adopted values of the rock shift and caving angles should conform with the geological section of rocks and are ordinarily fixed on the

basis of long-term observations, or by analogy with other regions. Rock displacement angles, marked on sections across the strike as β and γ, in practice are subject to variations, in soft rocks from 30 to 50° (more often 40-45°) and in harder and stable rocks from 55 to 80° (more often 60-65°). In mantle rocks, or drift ground these angles are set at 37-45° (dry rocks), 25-37° (moist and wet strata) and 20-30° (very wet strata). Rock caving angles, designated similarly to the displacement ones, usually exceed the latter. To choose

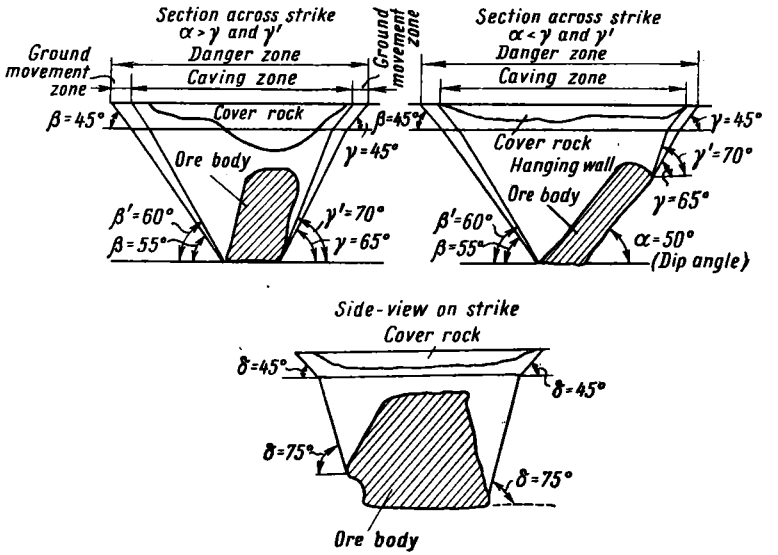


Fig. 34. Plotting boundary lines of shift and caving zones

a safe site for the surface structures a protective berm from 10-20 to 50-60 metres and more in width (depending upon the thickness of the ore body, depth of mining and the cost of the protected units) is traced around the danger zone bounds. The shaft house and surface plants should be located outside the berm. It should be borne in mind that there is a safe depth below which ground movements over the mined-out area do not affect the ground surface. Then there is no need to delimit any danger zone on the surface or construct protective pillars for surface structures. The safe depth is determined by multiplying the minable thickness by the safety factor with due account for the thickness of the ore body. The safe depth is measured in the vertical plane. The adopted value of the safety factor depends upon the thickness and the angle of dip of the mineral body, the priority of the surface plant protection, the composition and structure of rocks, the presence or absence of mine-

fill in the worked-out space. For example, under the conditions prevailing in the Donets basin the safety factor for first-priority structures is set within the range from 300 to 500 (in mining without backfill).

Construction of Safety Pillars

A safety pillar is plotted in horizontal and vertical projections on strike and across the strike by drawing the contours of the protected structures, protective berm and rock dislocation planes at definite angles depending upon the nature of the rocks, dip angle

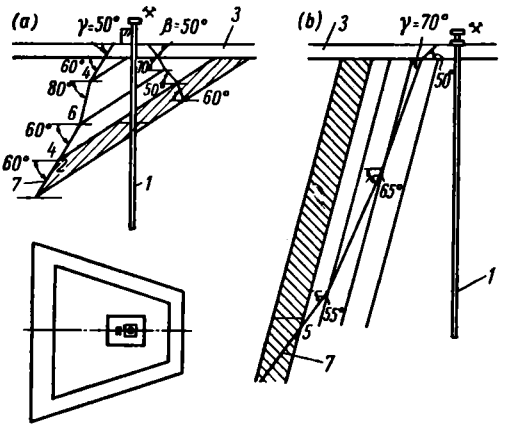


Fig. 35. Construction of a safety pillar:
a—in a gently sloping bed (section across the strike); *b*—in a steep bed; *1*—shaft; *2*—ore body; *3*—capping; *4*—shale of medium strength; *5*—shale of low strength; *6*—strong quartzite; *7*—safety pillar boundaries

of the ore body and the protection priority. At mining enterprises the protected structures are usually subdivided into three categories with regard to protection priority. The first category covers such structures and plants whose destruction may result in mass accidents, stoppage of operations, or cause considerable material losses.

In constructing safety pillars use is made of rules set up for each mining region by the Mining Inspectorate. Figure 35 exemplifies the construction of a safety shaft pillar across the strike in mining one of the ore deposits. Safety pillars are plotted in three projections with due consideration for protected structures and protective berms. The pillar outlines are plotted at different angles depending upon the nature of rocks. The bounds of an inclined shaft safety pillar are indicated by lines running parallel to the shaft, the total width of the pillar being at least 20-30 m.

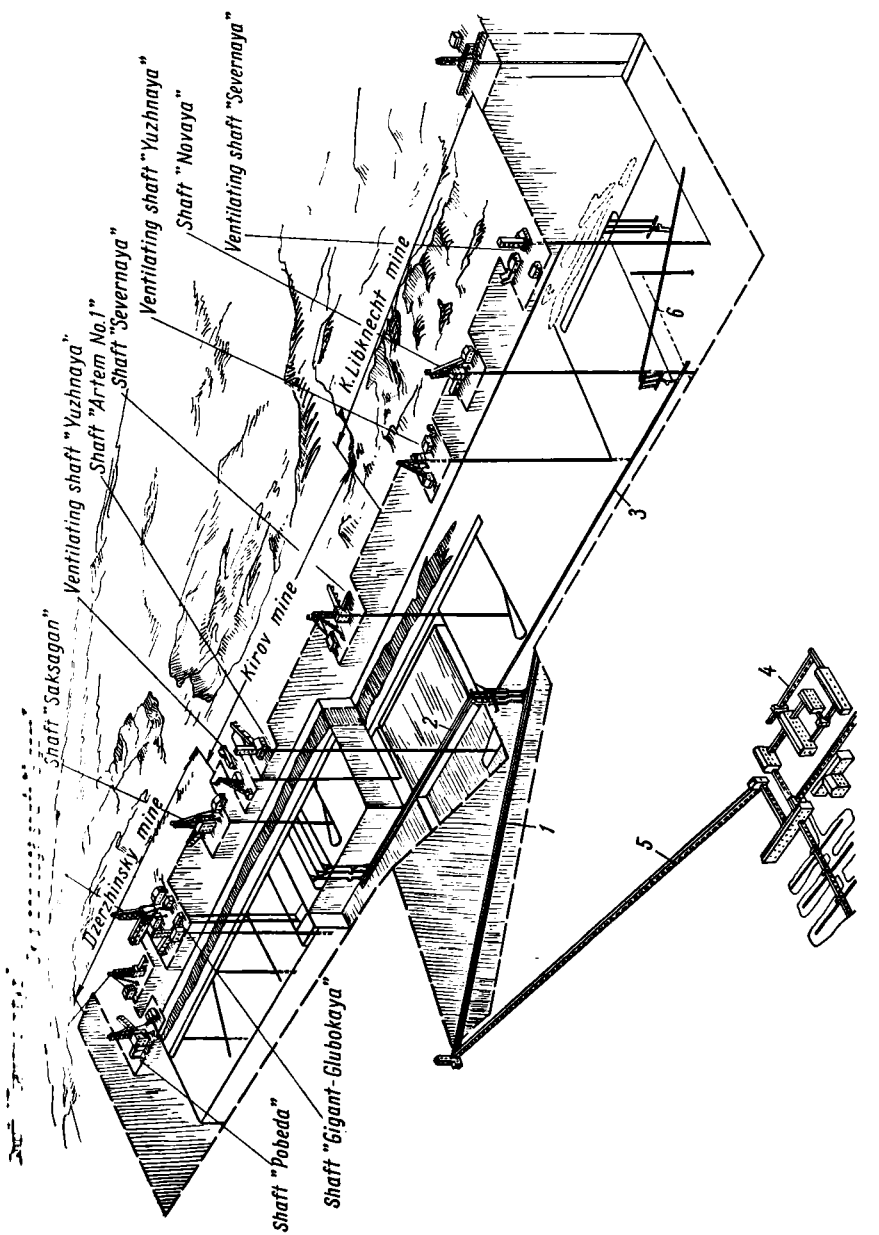


Fig. 36. Diagram illustrating integration of mine-fields of three mining enterprises in the USSR:
1—inclined shafts of Artem mine; *2*—460 m level conveyer-haulage drifts for transfer of ore from Dzerzhinsky mining enterprise; *3*—640 m level conveyer-haulage drift for transfer of ore from K. Libknecht mining enterprise; *4*—Southern ore preparation plant; *5*—conveyer gallery; *6*—inclined blind shaft

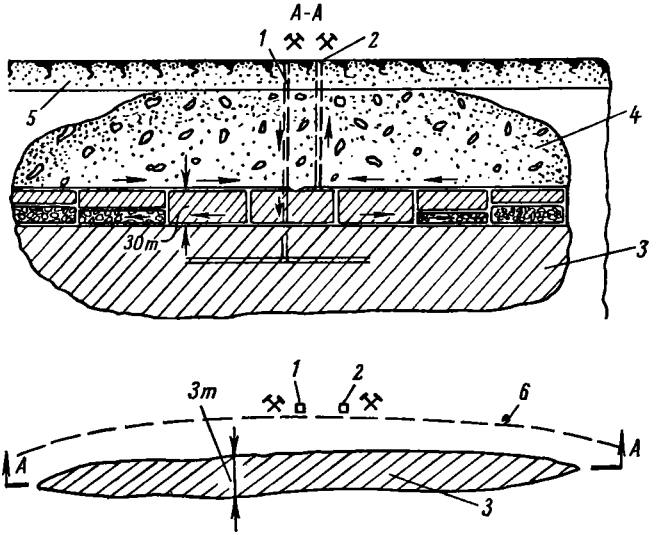


Fig. 37. Central shaft location in mining a steep ore body (indicating the direction of air current during extraction of one of the levels). Shafts are sunk in foot wall outside rock-shift zone:

1—hoisting shaft; *2*—ventilating shaft; *3*—ore masses; *4*—worked-out and packed space; *5*—capping; *6*—boundary of rock-shift zone; arrows indicate direction of air current

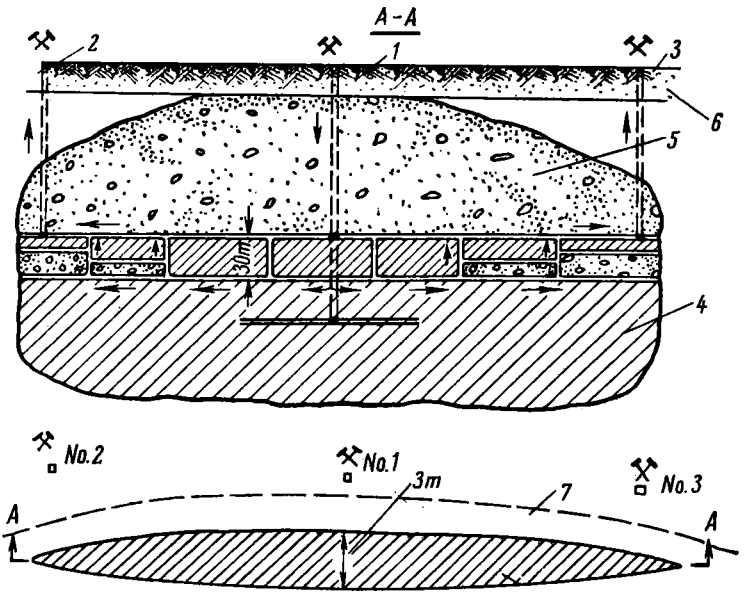


Fig. 38. Disposition of main shaft in the centre and of ventilating shafts at the flanks in mining a steep ore body (indicating the direction of ventilating air current during extraction of one of the levels):

1—hoisting shaft (in the centre); *2* and *3*—ventilating shafts (at the flanks); the shafts are sunk in foot wall outside rock-shift zone; *4*—solid ore in place; *5*—worked-out and packed space; *6*—capping; *7*—boundary of rock-shift zone; arrows indicate direction of air current

Size of Mine-Fields and Shaft Location

Ore deposits, which are usually of limited size, commonly have a single mine-field.

Ore deposits of sizable proportions are broken up into several mine-fields to ensure proper extraction of the mineral. Deposits are very seldom divided into mine-fields along the dip, therefore this arrangement is not discussed here. Ore occurrences are ordinarily mined through a single shaft until the ore body pinches out

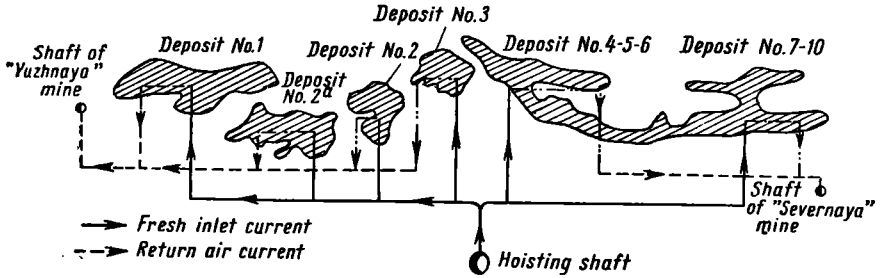


Fig. 39. Schematic layout of shafts in mining an intricately shaped occurrence consisting of several deposits

on the dip. The size of the mine-field along the strike depends upon a great number of technical and economic factors. In practice, the length of mine-fields in ore deposits is commonly set within the range from 400 to 2,000 m; in individual instances mine-fields as long as 5 kilometres and more are encountered. Mine-fields extending up to 4-5 km are adopted in occurrences of sizable proportions containing considerable ore reserves. Under favourable conditions of mining an extensive length of the mine-field makes it possible to cut down overall expenses per ton of extracted ore.

A project for integrating the mine-fields of three mining enterprises is under way in the USSR (Fig. 36). The maximum annual output of the combined enterprises will be 30 mil. tons.

Two types of mutual arrangement of the main and ventilating shafts are used in mining ore deposits: central disposition of the main and ventilating shafts and central disposition of the main shaft with the ventilating shafts at the flanks (Figs. 37, 38, 39).

Where ore bodies are of limited size and irregular shape the mine-fields in working ore deposits are made to assume a shape conforming to the outlines of the mineral occurrence (Fig. 39).

In low-output mines of limited mining depth shallow pits may be used instead of ventilating shafts. As mining operations progress downwards the pits are connected with ventilating raises driven in subjacent levels.

Early Development Methods

It is expedient to classify the methods employed in early development of ore deposits by the type of major openings with due consideration for their location in respect to the ore body (in the deposit itself, in the foot- or hanging wall).

Listed below are early development methods as classified by M. Agoshkov.

I. *Simple methods.*

1. Opening-up through an adit along and across the strike.
2. Opening-up through vertical and inclined shafts.

II. *Combined methods.*

3. Opening-up through adits with change-over to blind vertical and inclined shafts.
4. Opening-up through vertical and inclined shafts sunk from the ground surface with change-over to blind vertical and inclined shafts.

The principal methods for opening-up ore deposits include (Fig. 40*a*, *b*) early development (*a*) through a vertical shaft sunk in the foot wall of the deposit and (*b*) through an adit.

The other methods of opening, used less frequently, are (Fig. 40*c-g*): through an inclined shaft sunk in the ore or at a short distance from the ore body; through a vertical shaft sited at the flank of the deposit; through a vertical shaft in the hanging wall of the deposit; through a vertical shaft traversing the deposit; combined methods.

In opening-up through shafts, crosscuts are driven in each level, or one crosscut is made for several levels (group, or mother crosscuts). At the Krivoy Rog mines opening-up with twin levels has been practised widely since 1957-1958. Early development with twin levels increases the life-time of the main level and cuts down the volume of permanent workings. On the intermediate level a single-track opening of small cross-section and a shaft station with workings of minimal size are excavated. The expediency of early development with twin or triple levels can be ascertained by making technical and economic calculations.

Opening-up through a vertical shaft in the foot wall of the deposit, outside the rock-shift zone, is one of the most frequently practised methods in the early development of ore deposits with a dip angle in excess of 40°.

Opening-up through an adit is used in areas with mountainous relief. The adit may be driven in the deposit itself or across the strike, in the hanging or foot wall of the occurrence.

The advantages of opening-up through an adit were mentioned in the section dealing with the early development of bedded deposits and refer just as much to the opening of ore deposits.The major

advantage of opening through an adit is the absence of mine pumping charges. Approximate lengths of adits driven by using the money saved on mine pumping, in relation to the water lift height and the rate of its inflow, are shown in Table 4.

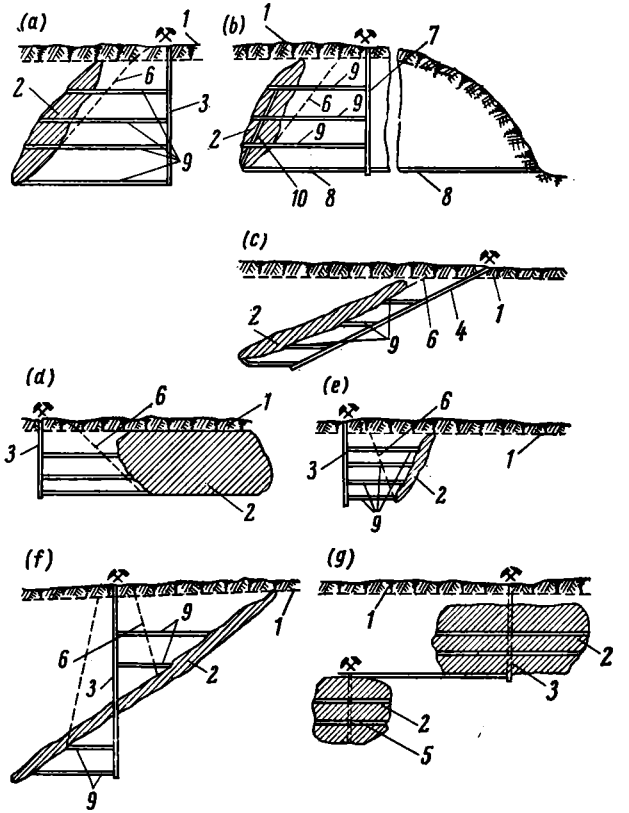


Fig. 40. Opening-up of ore deposits:

a—opening-up through vertical shaft sunk in foot wall; *b*—opening-up through adit; *c*—opening-up through inclined shaft in foot wall; *d*—opening-up through vertical shaft at the flank of deposit; *e*—opening-up through vertical shaft in hanging wall; *f*—opening-up through vertical shaft across the deposit; *g*—opening-up through a shaft sunk from the ground surface and a blind shaft at a deeper level; *1*—capping; *2*—ore body; *3*—main shaft (vertical); *4*—main shaft (inclined); *5*—blind shaft; *6*—boundary of rock-shift zone, or of protective pillar; *7*—service shaft; *8*—adit; *9*—crosscuts; *10*—ore chute

Adits are often used in mining ore deposits—copper, lead-and-zinc, etc.

Below we present brief information on opening-up by means of long adits in one of the lead-and-zinc and one of the copper deposits of the USSR.

Table 4

Length of Adits Driven by Using Money Saved on Mine Pumping with Different Water Lift Height

Shaft depth (water lift height), m	Water inflow, m^3/hr		
	20	100	300
	Adit length, m		
30	150	750	2,250
90	400	2,000	6,000
300	1,200	6,000	18,000

Example 1. *Opening-up of a lead-and-zinc deposit.* The deposit represents a composite lead-and-zinc-bearing pay streak extending for about 2 km, dipping at an angle of 85°, its thickness varying from 0.08 to 4 m and averaging 0.8-1 m. The occurrence has been opened up through a number of adits, one of them permanent. This double-track adit extends for 3,940 m. The cross-sectional area of its rough concrete-supported segments is 8.7 sq m and that of the finished segments 6.4 sq m; in timbered parts the rough section is 8.9 sq m and the finished one 6.74 sq m. The adit has undercut the main reserves of the deposit (six levels), simplified the transport of ore to the concentration plant and replaced a suspension cableway 4.5 km long. This considerably increased the efficiency of the mine and reduced the per-ton cost of the ore.

Example 2. *Opening-up of a copper deposit.* The deposit is a thick chalcopyrite lens occurring at a contact with keratophyre and diabase. The length of the lens along the strike is 460 m, the width 30-180 m and the angle of dip 45-60°. An adit 1,619 m long and 10.5 sq m in cross-section has undercut sizable ore reserves at the 373 m level, connected the deposit with the crushing-and-screening plant and made it possible to dispense with a complex and costly pumping installation (the mine water contained a considerable proportion of sulfuric acid).

Particular emphasis should be made on the part played by adits in exploring and prospecting metalliferous deposits. In an area with mountainous relief several adits may be driven, some of them intended for prospecting only, while the others are ultimately employed for mining.

In mining metalliferous deposits, inclined shafts are used in gently dipping ore bodies (less than 40°) when opening-up through a vertical shaft requires driving long crosscuts.

Inclined shafts are sunk in the deposit itself or, more often, in the foot wall rocks. A condition to be met in sinking inclined shafts

in the deposit itself is a regular mode of occurrence: the absence of faults and of pitch variations in the ore body, and also the low value of the ore abandoned in the protective shaft pillar. The boundaries of the protective pillar in an inclined shaft driven in the deposit proper are defined by lines drawn parallel to the shaft so that the pillar has a width of 20-30 m.

Inclined shafts sunk in the deposit proper allow supplementary exploration of the ore body to be made before stoping operations are begun.

Opening-up of deposits through inclined shafts is practised at a number of mines in working copper, lead-and-zinc, iron, gold and other ore deposits.

Inclined shafts may be widely employed in opening-up of deep levels, when vertical shafts equipped with skips have a limited hoisting capacity. The efficiency of inclined shafts equipped with conveyer transport facilities may be very high. This is exemplified (Fig. 41) by an all-round opening-up of deep-lying levels of the Kirov mine in the Krivoy Rog basin (down to a depth of 775 m and below) which is effected through two centrally-twinned inclined shafts driven at an angle of 16° and provided with conveyer transport facilities (two lines). The efficiency of one conveyer line reaches 5 thousand tons per hour. For ancillary duties a vertical shaft is being sunk.

To secure facilities for hoisting from a depth of 1,150 m it would have been necessary to sink four vertical skip-and-cage shafts equipped with 50-ton skips. The volume of ground excavated in making permanent mine workings would then have been 1.75 times as great as in opening-up through two inclined shafts. With further progress of mining in depth it would have been necessary to add one vertical shaft for every 250-300 m so as to compensate for the decline in skip hoist efficiency with depth.

Opening-up through inclined shafts will bring labour efficiency per miner up to 15.2 tons per shift. Moreover, this ensures independence of the capacity of the hoisting facilities from the mining depth and provides for the maximum possible degree of automation at this enterprise.

Among the combined methods of early development wide use is made of opening-up of the upper portion of the occurrence through a shaft and of the lower one (with the ore body flattening out on lower levels) through a blind shaft; opening-up of the upper portion of the deposit through an adit and of the lower one through a blind shaft (when the adit may undercut the up-dip portion of the deposit); opening-up of the lower levels of the deposit through a blind shaft, when ore reserves in the lower levels are limited and the crosscuts are to be of considerable length (Fig. 40 *g*).

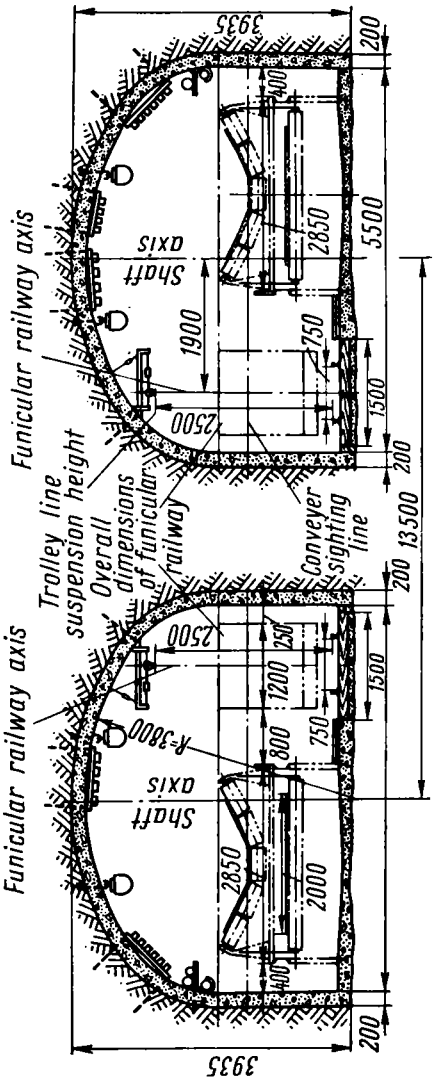


Fig. 41. Section of inclined shaft at the Kirov mine

The expediency of opening-up lower levels through a blind inclined shaft is confirmed by technical and economic calculations.

There are some other methods of early development, but these are applied rather rarely and their suitability should be established in each individual case by comparing them with the basic versions of opening-up ore deposits.

Of the new methods of early development used at mines abroad mention should be made of the following:

(1) Through inclined shafts fitted out with conveyer lines for hoisting and transportation of ore along haulageways (England, USA).

(2) Through inclined shafts with conveyer hoisting plants and electric locomotive transport along haulageways (Canada, Lorraine in France, USA).

(3) Through inclined shafts with conveyer hoisting and self-propelled transport along haulageways (Canada). For the movement of self-propelled equipment a special inclined shaft driven at an angle of 12° is provided which communicates with production levels.

(4) Opening-up of the central section in a thick deposit (at a Swedish iron ore mine with a production capacity of up to 15 million tons per year) through eight skip shafts and one service cage shaft in the foot wall. A single shaft house accommodates 9 hoisting plants, bins for crushed ore delivered from underground workings, screening units and machinery of a magnetic separation plant with bins for concentrates. Each of the haulage levels (the level interval height being 50 m) is provided with a two-stage crusher. Each pair of shafts is serviced by a crushing unit of its own with a production capacity of 1,000 tons per hour. From the crusher the ore is fed to one of the bins of 1,000-ton capacity each. The ore is delivered to the crushing plants in 20-ton side-dump cars drawn by electric locomotives. With the system of sublevel caving employed extraction of ore in sublevels is done with the use of self-propelled hauling units flitted down from the ground surface along a special slope communicating with individual sublevels. In new levels the interval height is increased to 100 m, and a change-over to 25-ton ore cars and an electric locomotive with adhesion weight of 40 tons is envisaged.

(5) Opening-up through vertical shafts with the ore transferred down to storage levels via permanent stepped ore chutes (Canada). Storage- or gathering levels are spaced at 300-400 m and are equipped with crushing plants.

Among the above-discussed variants the most promising is early development through inclined shafts fitted out with conveyer lines; an ancillary slope is made for the movement of self-propelled (drilling, loading and transport) equipment.

In opening-up through deep vertical shafts it is expedient to make storage (gathering) levels with crushers and with the transfer of ore along permanent ore chutes from the intermediate levels down to the main level. As distinct from sheet deposits sinking of shafts involving abandonment of protective pillars is a relatively rare practice in the mining of ore deposits, since it is uneconomical to abandon considerable reserves of frequently very valuable ore in safety pillars. These are left when it is impossible to choose an appropriate site for the sinking of a shaft outside the rock-shifting zone, or in mining thin occurrences of sizable extent both on the strike and the dip.

Early Development at Great Depth

At present winning of ore in Soviet mines is mostly done at a depth of 300-500 m, and only in a few mines at 700-800 m. In the next 20 years, however, the mining depth in a number of mines will reach 1,000-1,200 m. Some mines abroad are worked at a very great depth. Thus, for example, a mine in India operates at a depth exceeding 3,200 m, and 40 mines in the Republic of South Africa, at depths ranging from 1,500 to 3,800 m.

In a number of Canadian mines stoping is done at a depth of over 2 km.

When erecting mine structures account should be taken of flattening out of zones of rock-shift in the hanging and foot walls. In some of Krivoy Rog mines rock-shift angles at a depth of 350-400 m have diminished to 55-57° in the hanging wall rocks and to 33-40° in the foot wall rocks.

In mining ore deposits with rigid rocks at great depths rock shocks and bursts may occur due to overstrain of the rock under the effect of high ground pressure.

Rocks with an ultimate resistance to crushing below 800 kg/cm^2 at depths below 800-1,000 m exhibit plastic properties (creep), even such rocks as sandstones becoming plastic. In selecting modes of early development at deep levels no safety pillars should be left. Timbering in deep levels of sedimentary deposits should be strong and yieldable. Frequent use is made of steel support with reinforced concrete tie-beams and compact filling of the space between support and the walls of the working.

Ventilation and cooling of the air in deep horizons should be calculated on the basis of maximum permissible parameters of the mine climate: air temperature of 26.5-27°, humidity of 80% and air currents velocity of 4 m/sec. (*Investigations of the Mining Research Institute of the Ukrainian Academy of Sciences.*)

The air velocity may be raised above 4 m/sec by sending air currents along special workings not to be used for the passage of men. Below is a brief description of the opening-up of gold and uranium occurrences in the Republic of South Africa, mined at a great depth. The upper levels of the deposits were formerly opened up through multistage vertical shafts with ancillary inclined shafts. Of late, due to technical achievements in the field of mine hoisting (use of multirope hoists with tower headframes) the mines have switched over to single-stage hoisting of ore via vertical shafts extending down to 2.2 km by using twin crosscuts to allow for the passage of large volumes of air. Owing to increased rock pressure the main drifts are as a rule driven in country rocks (stone drifts). The mine-field is opened through a series of shafts, each one of them having definite functions (hoisting of ore, delivery of gangue, inflow of fresh and outflow of polluted air). The shafts are located outside the area affected by stoping operations. The ore is passed along permanent ore chutes down to storage levels (servicing 4-5 ordinary levels) with crushers installed there. A combined ventilation scheme (forced and exhaust aeration) with wide utilization of axial-flow fans is widely used. The air temperature at the faces is maintained at 27-28° by means of refrigeration plants.

3-2. DIVISION OF MINE-FIELDS INTO LEVELS OR PANELS AND THE ORDER OF THEIR EXTRACTION

In working ore deposits mine-fields are divided into levels or panels just as in mining bedded occurrences.

Level interval height. In mining ore deposits the vertical level interval height varies from 20 to 100 m (more often 40-60 m). The choice of the level interval height is mainly influenced by: the nature of the deposit, its size and degree of exploration, the angle of dip, the rigidity and stability of the ore and enclosing (host) rocks, the adopted method of early development and the system and intensity of mining. An increased level interval height makes it possible to reduce the scope of development work per ton of extracted ore and the amount of ore abandoned in pillars, whose robbing entails considerable complications and excessive losses.

The factors limiting the level interval height include: increased costs of delivering supplies and equipment to production levels, difficulties encountered in upraising and maintenance of block ascending workings, waste of workers' energy in climbing up and down the workings.

Furthermore, one of the major arguments against the adoption of a large level interval height is the rise in the pressure of wall rocks when they are exposed over a considerable area, this affecting

the safe conduct of operations. The optimal level interval height can be determined in each individual case by technical and economic calculations.

An optimal level interval is one at which capital outlays and running expenses (which depend upon this interval) are reduced to a minimum.

At the same time the correct level interval should ensure safety of mining operations, the possibility of early preparation for stoping, and also reduction of over-all losses and dilution of the ore. Because of relatively low expenses to be considered in determining the level interval (not exceeding 10% of the total mining cost of the ore) and small variations of these expenses with differing level interval height (according to investigations of M. Agoshkov, Corresponding Member of the Academy of Sciences of the USSR) the latter should be chosen within the range from 40 to 100 m with due regard for the existing mining and geological conditions. In setting the final level interval height consideration should be given to a possible division of a horizon into individual sublevels and to the expediency of providing lifts for men, supplies and equipment in each group of blocks, which will permit the level interval to be increased and the associated expenses reduced.

Level interval heights adopted in practice with the various mining methods are indicated in the description of each particular method.

A novel trend in the level-wise exploitation of a mine-field is its extraction by twin and triple levels, the ore being transferred from the upper levels down to the lower ones through permanent ore-chutes and then broken by underground crushing units. This simplifies the development of intervening levels (Fig. 42).

As a rule, horizons are worked in descending order. The number of simultaneously mined levels varies from one to three or four. In mining ore deposits multilevel extraction is practised rather frequently due to increased demands on the intensity of ore extraction in deposits of limited size. Not all mining methods allow for simultaneous stoping in several levels. This entails increased expenses in servicing the levels. The service life of a level usually does not exceed 3-5 years, which is much less than in the case of coal mining. For this reason all level workings which are normally excavated in sufficiently rigid and stable rock, are either not supported at all, or have timber support. In the practice of mining ore deposits concrete support of lateral openings is used comparatively seldom. In recent years wide use has been made of pre-fabricated reinforced and suspension roof support (rock bolting). Pre-fabricated reinforced concrete support is practical even in workings with a limited service life.

As far as suspension roof support is concerned the most promising of the known designs are reinforced concrete roof bolts, or rock bolts. Such bolts ensure very strong support both in soft and rigid rocks, they cost much less than steel ones, the process of fixing

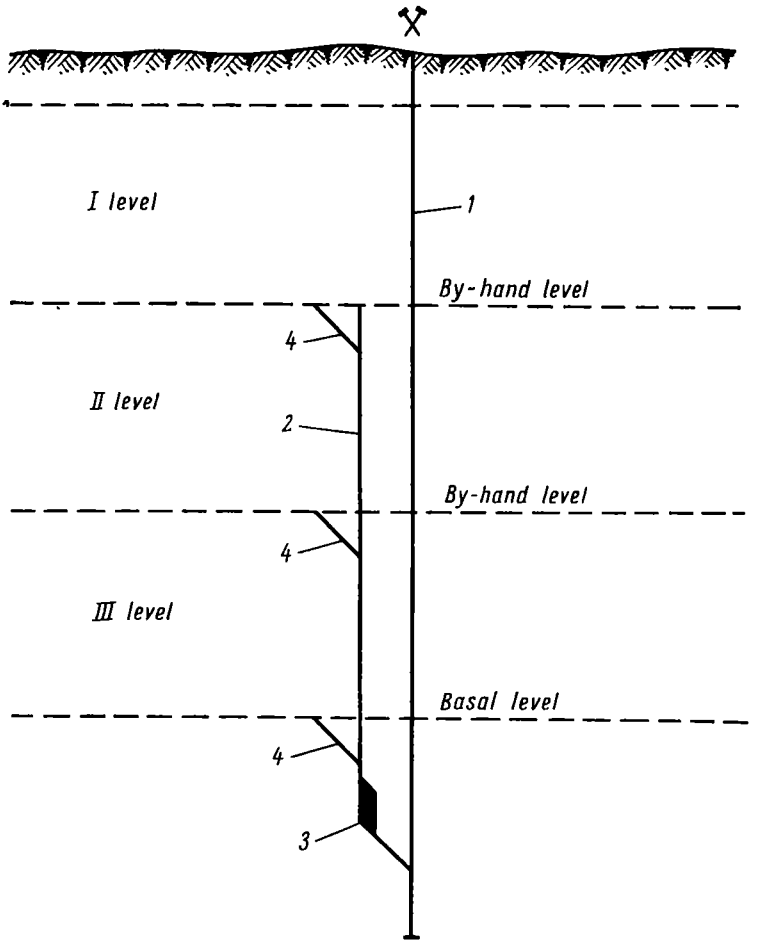


Fig. 42. Opening-up with triple levels (vertical section):
1—main shaft; *2*—main dumping chute; *3*—crusher chamber; *4*—inclined descent

is simpler, they are reliable in exploitation and require no control. Reinforced concrete bolts are formed by filling a borehole with a cement-water or cement-sand grout and subsequent insertion

of metal reinforcement. The grout is made of high-quality quick-setting cements grades 400-600. Sand or granulated slag (a mixture of cement and sand in a proportion of 1 : 2) is used as aggregate.

Reinforcement may be in the form of wire (d = 4-10 mm), round steel rods (d = 10-30 mm), or old steel rope. To fill boreholes with a cement-sand grout, mobile pneumatic feeders of various designs are used. To preclude rock inrushes, a net made of wire (d = 2-10 mm, 20 × 20 mm, to 100 × 100 mm mesh) is fixed between the rock bolts. The net is pressed against the rock with thrust plates or bolt catches and in the case of a long service life has a gunite covering. The walls of the workings are then also covered with gunite.

As regards the disposition of the stoping front relative to the shaft, the level may be mined on both sides (bilateral extraction) or to one side of the shaft (unilateral extraction).

Unilateral extraction is inadvisable because of a considerable haul length, but it cannot be avoided where it is difficult to locate the shaft centrally.

Division of a level into blocks and the order of their mining. In mining ore deposits a level is more often than not divided into extraction areas called blocks. The general order of operations, the direction of mining and the size of blocks are contingent upon the adopted method, mining conditions, output and the date fixed for commencing exploitation of the level. In working ore deposits the extraction of blocks in a level on the strike may be done as follows: in the direction from the hoisting shaft to the mine-field boundaries (advance order); from the latter towards the hoisting shaft (retreat order—see Figs. 37, 38); by using a combined method; by two-stage mining in individual producing sections; by stoping along the entire strike line.

In extracting an ore body starting from the shaft towards the mine-field boundaries the haulage drift is either pushed forward parallel with the progress of stoping (with a certain lead) or its drivage is completed prior to the starting of stoping operations. In working blocks from the mine-field boundaries towards the shaft the haulage drift is made before stoping is begun.

Both in advance and retreat mining of the level blocks are stoped successively one after another, or in groups of two or three. The advantages and shortcomings of advance and retreat mining have been discussed in the section on opening of bedded deposits. In combined extraction blocks are mined simultaneously from the shaft towards the mine-field boundaries and vice versa.

In two-stage mining some blocks are extracted first, the remaining ones coming next. This procedure is usually followed with mining

methods where some blocks (rooms) are extracted by one method and others (pillars) by a different one. Mining along the entire line of strike is done simultaneously in all blocks (during the stage of all-round development of stoping). This type of extraction is practised with ore bodies of very limited size, and also with some mining methods providing for the commencement of stoping along the entire line of strike (for example in top slicing).

Mining of blocks in a level down and up the dip. Blocks are mined either upwards or downwards. In some mining methods extraction of individual blocks is done along the strike simultaneously throughout the entire level interval.

The order followed in extracting blocks is considered in detail in the subsequent discussion of the mining methods employed in working ore deposits.

Preparation of blocks for stoping. Preparation of a block implies excavating a set of workings to enable stoping operations to be started.

In most cases development workings pierce the ore body itself (productive development), less frequently they traverse country rocks (stone development). Sometimes a level is prepared for stoping by simultaneous excavation of workings in the ore and country rocks (combined development).

The method for preliminary development should be chosen on the basis of technical and economic comparison, due account being taken of the service life of the level, ventilation and transport conditions and the requirement for the reduction of ore losses. Stone development increases the cost of driving the workings, but permits one to reduce expenses for their upkeep, improve haulage and ventilation conditions and decrease ore losses in the pillars.

In addition to workings of the main level (drifts, crossdrifts) and to raises other openings are excavated during preliminary development of a block, whose disposition is indicated in the discussion of individual mining methods.

The scope of development work may be characterized by two factors:

(a) The volume of ore taken out during excavation of development workings in per cent of the total minable reserves in a level or block (it varies from 2-3 up to 10-15%).

(b) The amount of ore per 1 m of development workings. As distinct from the first point, this covers both productive and stone development.

Breaking up a mine-field into panels. In mining ore deposits panelwise extraction (panelling) is practised in flat or gently sloping ore occurrences (with a dip of up to 10°). A panel is broken up into individual producing sections called blocks. A typical example

of panelling is the practice adopted at the Dzhezkazgan mine (Fig. 43).

The length of the panel should accord with the size of the mine-field, the width with the type of equipment used: with slusher haulage it is up to 50 m, and with self-propelled vehicles up to 100-150 m. In working out thick deposits the airway is driven in their back

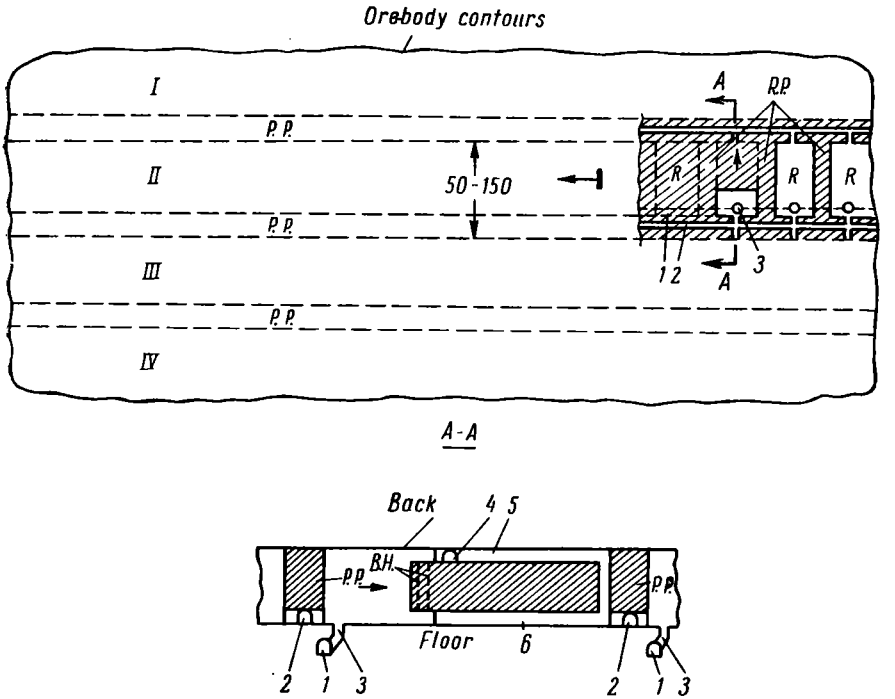


Fig. 43. Layout of panels in mining a flat ore body with drivage of lateral (stone) haulageways (plan view) :

R—room; *R.P.*—rib pillar; *B.H.*—borehole; *I, II, III, IV*—panels; *P.P.*—panel pillar; → general direction of panel extraction; → direction of room extraction; *1*—lateral haulage drift; *2*—connecting reef drift; *3*—ore-chute; *4*—vent drift; *5*—upper undercut opening; *6*—auxiliary working

(along the major axis of the panel). During the preparation of a panel a stone or reef drift is cut. With rail transport, when the bottom of the ore body is uneven, a stone haulageway is driven. Self-propelled vehicles permit greater gradients of the roadways, and in this case productive development is practised.

3-3. ANNUAL PRODUCTION CAPACITY AND SERVICE LIFE OF MINES IN WORKING METALLIFEROUS OCCURRENCES

General

The annual output of a mine is one of the basic factors in the work of mining enterprises. The prime cost of the ore, capital investments, the scope of mining and construction work, time-schedule for commissioning and many other factors depend upon the planned production capacity of the mine.

Under the conditions prevailing in Soviet industry fixing of the yearly output for a mining enterprise is closely connected with requirements associated with the State Plan for the Development of National Economy and the planned need for a particular mineral, which take account of the interests of the national economy of the USSR as a whole. The annual output of a mining enterprise is usually determined by the long-term program for the development of the corresponding industry. The production capacity of a mining enterprise should be fixed when determining the output of individual enterprises in harmony with the pre-set production figures for the region or group of occurrences being mined, in deciding upon the priority and number of simultaneously commissioned production units during commercial exploitation of big, formerly unmined deposits. In all instances the planned production capacity should be in keeping with maximal technical and economic expediency.

In practice the adopted yearly output for a mine should take into account the following factors: (1) reserves of useful components available in the deposit being mined; (2) practical combination of sinking-fund and running expenses; (3) size of the mine-field; (4) exploitation possibilities with the contemplated mining methods. Some of the above factors are considered in laying down the yearly output of the mine with special reference to mining potentialities (the first, third and fourth factors), the others—according to the economic requirement for the minimal first cost per 1 ton of ore throughout the service-life of the mine (the second factor). The demand of minimal mining cost also implies the least possible labour expenditure.

In addition to the above-listed factors one must take account of the degree to which the deposit has been explored and of prospects for an increase in its reserves. Due consideration should also be given to reaching the target output figures within a short period of time, to the possibilities of ore dressing, etc.

Approximate Estimation of the Annual Mine Output According to Mining Potentialities

The annual mine output according to mining potentialities can be estimated with a varying degree of accuracy. In assessing deposits under exploration and correlating the exploration results with subsequent construction and exploitation of the mine approximate methods for fixing its yearly output can be used.

One of the approximate methods for estimating the annual production capacity of a mine uses the exploitation or utilization factor, another the annual downward advance of stoping.

Using the exploitation factor, the annual mine output is estimated according to the formula

$$A = S\eta$$

where A = annual production of the mine in tons;
S = average area of the ore body mined, or of several bodies, sq m;
η = exploitation factor—tons of ore obtained annually from 1 sq m of the ore body area worked. This figure is based on the experience gained in mining analogous deposits; it varies within the range from 30 to 120 t/m² per year, depending upon mining conditions and the method employed.

Determination of the mine output by using the exploitation factor yields very approximate results, since here its value is based on practical data on the mining of other occurrences, and there can be no complete similarity.

The second approximate method of estimating the annual mine output is more accurate, it is based on the annual downward advance of stoping operations. This procedure has been proposed by M. Agoshkov, Corresponding Member of the Academy of Sciences of the USSR. The formula used is:

$$A = VS\gamma\frac{K_r}{1-K_d}$$

where A = annual output of the mine, tons;
V = mean annual downward advance of stoping over the entire area of the ore body in metres per year (varies from 15 to 40 m per year);
S = average area of the ore body (or several ore bodies) being mined, sq m;
γ = volume weight of the ore, tons/m³;
K_r = coefficient of ore recovery; it depends upon the method used and mining conditions (usually varies from 0.80 to 0.95; in room-and-pillar methods of mining salt occurrences the value of K_r drops to 0.6-0.5);

K_d = ore dilution factor; it depends upon the method applied and mining conditions (usually fluctuates between 0.20 and 0.05).

In estimating the annual mine output by this method it is possible to take into account a number of points exerting an appreciable effect upon the mine output, the volume weight, losses and dilution of the ore, and also to set down the increase in depth of stoping in conformity with the adopted size of the mine-field and the number of levels worked simultaneously.

Values for annual downward advance of stoping are taken from Tables 5 and 6 compiled by M. Agoshkov on the basis of a great body of field data and numerous calculations.

Table 5

Values for Annual Downward Advance of Stoping, in Relation to the Mine-Field Size and the Number of Levels Worked Simultaneously

Category of mine-fields and number of levels worked simultaneously	Annual vertical downward advance of stoping (V), m		
	Mean values	Minimum values	Maximum values
Very large mine-fields (over 1,000-1,200 m with ore body thickness exceeding 15 m, and over 1,500 m with ore body thickness less than 15 m)			
single-level extraction	15	12	18
double-level extraction	18	15	20
Large mine-fields (from 600 to 1,000 m with ore body thickness in excess of 15 m, and from 1,000 to 1,500 m with ore body thickness of less than 15 m)			
single-level extraction	18	15	20
double-level extraction	25	20	30
Mine-fields of average size (from 300 to 600 m with ore body thickness over 15 m, and from 600 to 1,000 m with ore body thickness of less than 15 m)			
single-level extraction	22	18	30
double-level extraction	27	20	35
triple-level extraction	32	25	40
Small mine-fields (up to 300 m, with ore body thickness over 15 m, and up to 500-600 m with ore body thickness of less than 15 m)			
single-level extraction	20	15	25
double-level extraction	25	20	30
multi-level extraction	35-40	25	60

Table 6

Correction Factors K_1 and K_2 for the Annual Downward Advance of Stoping in Relation to the Dip Angle and Thickness of the Deposit

Angle of dip, deg	Correction factor K_1	Ore body thickness, m	Correction factor K_2	Angle of dip, deg	Correction factor K_1	Ore body thickness, m	Correction factor K_2
90	1.2	up to 5	1.25	45	0.9	15-25	0.8
60	1.0	5-15	1.0	30	0.8	Over 25	0.6

Values of V are indicated in Table 5 for occurrences 5-15 m thick, dipping at an angle of 60°.

For deposits of other thicknesses and angles of dip it is necessary to introduce correction factors K_1 and K_2 according to Table 6.

Table 5 includes average, maximum and minimum values of V, contingent upon the degree to which the deposit has been explored, the regularity of its occurrence, the tectonics and hydrogeology, the conditions of its opening, the rate of advance of development workings and the mining method employed.

Maximum values for V should be taken with a favourable, and minimum values with an unfavourable combination of factors.

The actual downward advance of stoping in iron mines of the USSR averages about 20 metres per year and in non-ferrous, rare-metal and gold mines 20-25 (maximum 30-40) metres per year.

The above-discussed approximate methods of estimating the annual production capacity of a mine are applicable to ore bodies dipping at an angle exceeding 30°.

Estimation of the Annual Mine Output Based upon the Conditions of Deployment of Stoping Operations

More accurate determination of the annual mine output can be achieved by carrying out appropriate calculation procedures. This method can be employed, for instance, in setting down the annual tonnage of mines working flat and gently sloping occurrences when no data for estimating the mine efficiency by the approximate method are available. Given below is brief information on the estimation of the annual mine output based upon the conditions of deployment of stoping operations.

With single-stage extraction the yearly mine tonnage can be estimated by this method according to the formula

$$A = \frac{Nnp}{\mu\left(1+\frac{\rho}{100}\right)} \text{ tons}$$

where N = number of blocks (extraction units) with due allowance for stand-by ones subject to simultaneous stoping;
n = number of simultaneously active stopes in block;
p = average yield of stope, tons/year;
μ = stoping recovery factor depending on mining methods in use and general mining conditions (it usually equals 0.8-0.95);
ρ = percentage of stand-by blocks with respect to total number of active ones (ρ may be roughly put down at 20-30% when the number of active blocks is up to 5, 15-25% with 5-10 blocks and 10-15% with more than ten blocks).

The yield of an active stope depends upon its breast area and advance rate. The former is conditional upon the design parameters of the mining method, the latter upon the equipment and organizational set-up.

In some instances it is simpler to estimate the annual mine output on the basis of the number of blocks stoped and their yield, then

$$A = \frac{NP}{\mu\left(1+\frac{\rho}{100}\right)} \text{ tons}$$

where N = number of production blocks including stand-by ones;
ρ = yield per block, tons/year.

The other designations are given above.

The annual production of a block in a stoping stage may roughly be determined on the basis of 300 working days in the year and a daily yield of a block ranging between 50 and 100 tons in working small veins and 600-1,000 tons or more in mining massive deposits by efficient methods. More precise figures can be obtained by appropriate calculations.

In determining N one should proceed from a normal situation in which one of the levels is in the stage of full deployment of stoping operations, the overlying level in the stage of their conclusion, and the subjacent one in the stage of development. As pointed out earlier, the above formula can be applied to the case of simple single-stage working of a block.

Blocks are often mined in several stages, for example, extraction of the first slice, room, back and bottom, robbing of the intermediate pillar. In this case the annual output of the mine should be estimated with due account taken of the number and yield of blocks going through different stages of mining.

In working small, thin and very thin deposits the annual mine output determined by calculations should be verified with reference

to the conditions of their development:

$$A_d = \frac{T_e}{t_d w}$$

where A_d = yield feasible with timely development of level, tons/year;

T_e = extractable ore reserves in the level with allowance for losses and dilution, tons;

t_d = time required to develop the level for stoping (time spent in opening-up of the level is not considered, since opening-up must proceed ahead of development), years;

w = coefficient allowing for advance of development against stoping (w is equal to 1.3-1.4); the level development time is estimated on the basis of the time needed to drive a level drift.

The time spent in raising uphill openings is not considered, since they can be driven simultaneously with the drift.

Estimating the Most Economical Annual Production Capacity of a Mine

In working mineral occurrences the final economic criterion is the prime cost of the mined ore which is made up of operational expenses and depreciation of capital investments (sinking-fund). The former amount to 80-90% and the latter to 10-20% of the total cost of the ore.

An analysis of operational expenses per 1 ton of ore shows that with rising production capacity of a mine a part of operational expenses (development work, stoping) remains unchanged, whereas the other part decreases, thus, on the whole, operational expenses per 1 ton of ore reduce. With a 10% rise in the annual output operational expenses per 1 ton of ore at big mines go down by about 1-1.5% and at small ones by approximately 2.5-3%.

With constant ore reserves in the mine-field, depreciation of the invested capital, in contrast to operational expenses, increases parallel with the increase in mine production, since in this case more powerful equipment and more expensive structures are needed.

A graph of changes in operational expenses and depreciation with special reference to one of the examples of mining practice is presented in Fig. 44. It shows that with varied mine performance there is a considerable region of minimum net cost beyond which the cost begins to climb up noticeably with rising mine output but this increase usually occurs outside the technical possibilities of mining the deposit.

As a rule, to minimum-net-cost regions there correspond mine service-life figures listed in Table 7.

Table 7

Service Life of Mines as a Function of Their Annual Output

Annual output, tons	Expedient service-life, years	
	Small depth, easy early development and stoping conditions	Great depth, difficult early development and stoping conditions
50-100	From 3-4 to 8-10	From 4-5 to 12-15
100-200	" 4-5 " 10-12	" 5-6 " 15-18
200-500	" 5-6 " 12-15	" 6-8 " 20
500-1,000	" 8-10 " 15-18	" 10-12 " 25
Over 1,000	" 12-15 " 20-25	" 15 to 30 and more

Economical service-life periods for mining enterprises. The service life of a mine is determined as the quotient of division of the mine-field reserves by the adopted annual production of the mine. In the majority of cases economical service-life periods for mines correspond to the data of Table 7. Shorter life spans may be adopted when there is a particularly great need for the given mineral, when it is a material of high value, and with prospects for an appreciable increase in the reserves of the deposit being mined.

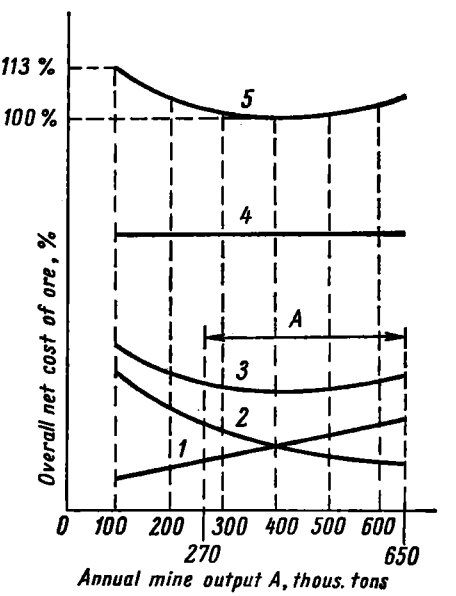


Fig. 44. Relationship between net cost of ore and mine output: *1*, *2*—variable components of net cost; *3*—sum-total of variable components of net cost; *4*—constant components of net cost; *5*—total cost per ton of ore; *A*—region of optimal mine output values (270-650 thous. tons per year) estimated to within 10%

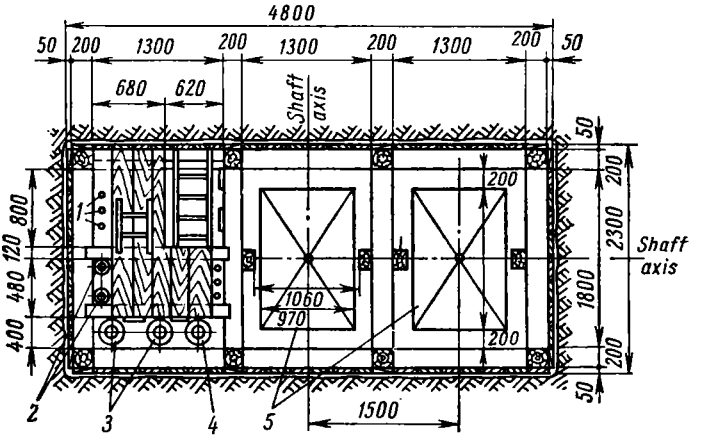


Fig. 45. Shaft with suspension timbering having annual output of 30-70 thous. tons of ore (finished section 7.74 sq m, overall rock section 11.2 sq m):

1—cables; *2*—water-supply pipes (d = 50 mm); *3*—mine drainage pipes (d = 100 mm); *4*—compressed air pipes (d = 100 mm); *5*—cage accommodating a mine car of 0.5 cu m capacity

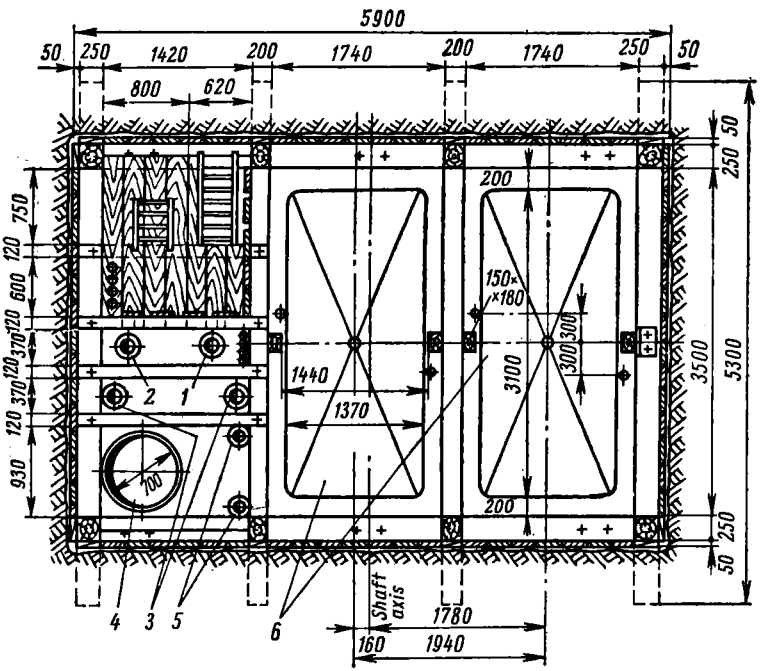


Fig. 46. Shaft with suspension timbering having annual output of 100-300 thous. tons of ore (finished section 18.6 sq m, overall rock section 24.2 sq m):

1—flow pipe (d = 150 mm); *2*—compressed air pipe (d = 150 mm); *3*—mine drainage pipe (d = 150 mm); *4*—ventilation pipe; *5*—water supply pipes (d = 100 mm); *6*—cages

3-4. EARLY DEVELOPMENT WORKINGS IN ORE DEPOSITS

Main Shafts and Level Mine Workings

Early development workings in metalliferous deposits often have a limited service-life and are made in stable rocks. For this reason they are usually supported by timber and their section is given the

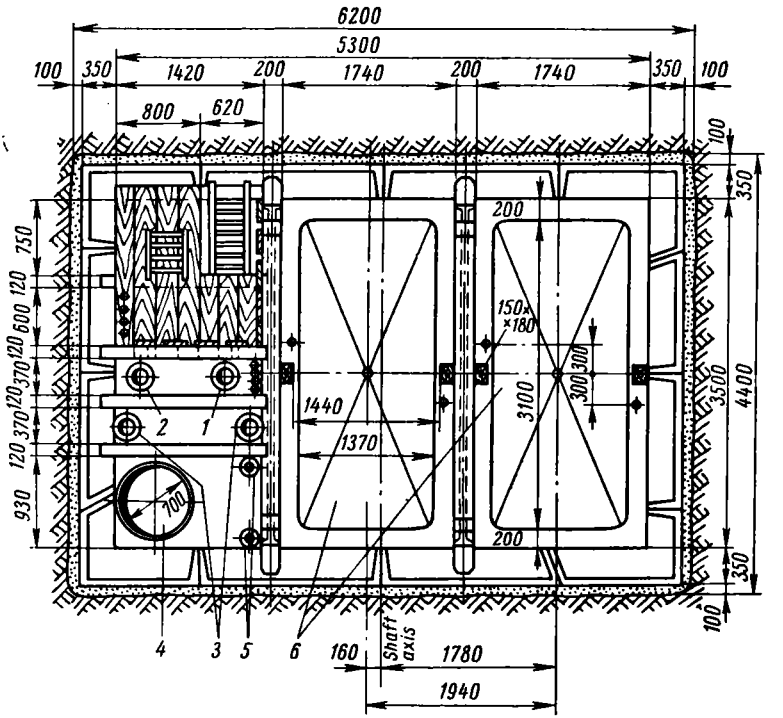


Fig. 47. Shaft with prefabricated reinforced-concrete support having annual output of 100-300 thous. tons of ore (finished section 18.6 sq m, overall rock section 27.28 sq m):

1—flow pipe (d = 150 mm); *2*—compressed air pipe (d = 150 mm); *3*—mine drainage pipes (d = 150 mm); *4*—ventilation pipe; *5*—water supply pipes (d = 100 mm); *6*—cages

corresponding shape. Shafts are generally made rectangular and are supported by suspension timbering of beams 200-300 mm thick with bearing sets spaced at 25-30 m (Figs. 45, 46). In unstable rocks shafts are supported by close-set timbering. Shafts designed for a long service-life or subjected to a high rock pressure are supported by concrete, prefabricated reinforced concrete or tubbings and are of rectangular or, more often, round shape (Figs. 47, 48).

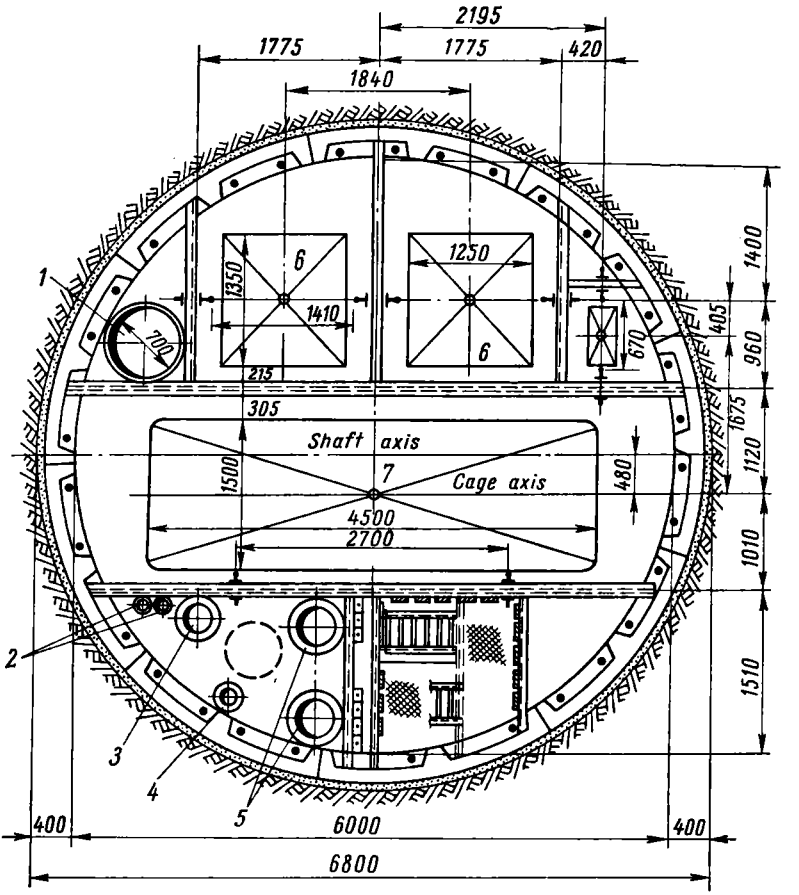


Fig. 48. Tubbed shaft with annual output of 300-600 thous. tons of ore (finished section 28.26 sq m, overall rock section 36.3 sq m):

1—ventilation pipe; *2*—water supply pipes (d = 100 mm); *3*—compressed air pipe (d = 250 mm); *4*—flow pipe (d = 150 mm); *5*—mine drainage pipes (d = 350 mm); *6*—skip of 2, 3 or 4 cu m capacity; *7*—cage

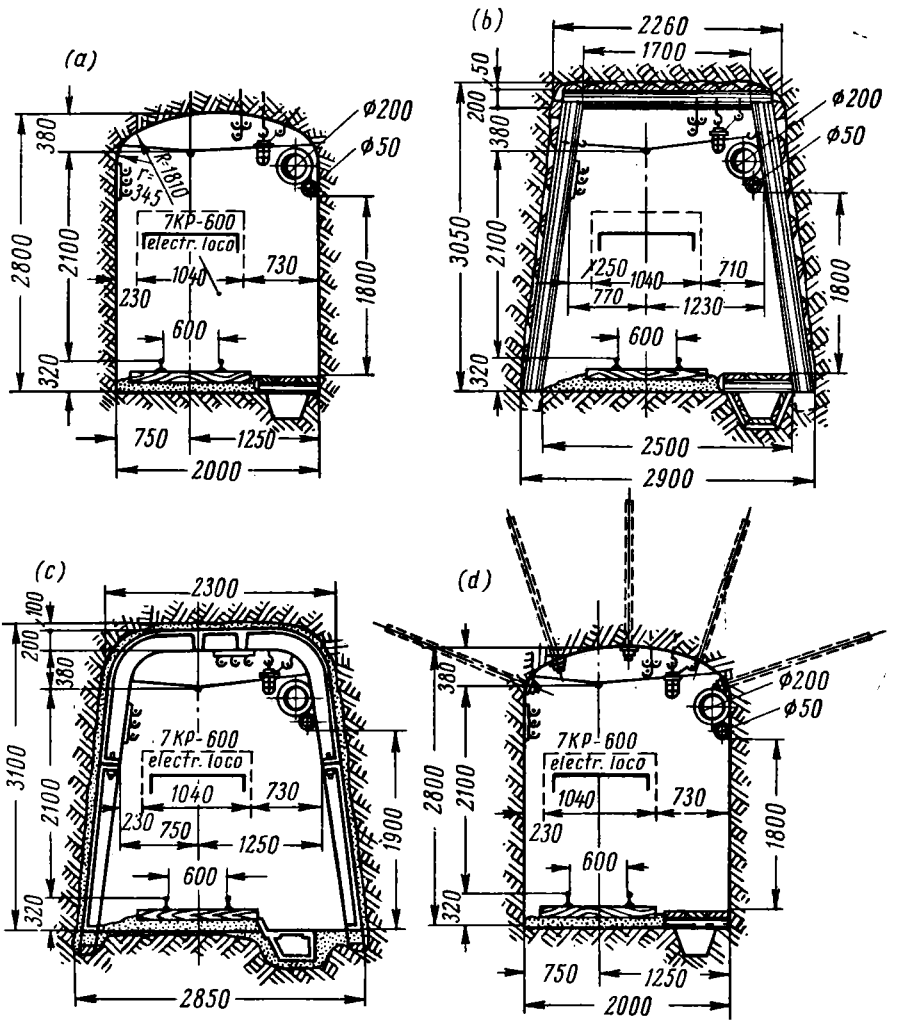


Fig. 49. Horizontal single-track workings in a mine with annual output up to 300 thous. tons provided with different types of mine support:

a—unsupported (overall rock section 5.5 sq m); *b*—timbered (finished section—5.4 sq m, overall rock section 7.9 sq m); *c*—with prefabricated reinforced-concrete support (finished section 5.4 sq m, overall rock section 8 sq m); *d*—with suspension roof support (finished section 5.1 sq m, overall rock section 5.5 sq m)

Level openings are supported by three-piece or, less often, four-piece sets, and have a trapezoidal section. In rigid stable rocks level openings are driven without artificial support or with roof bolting (suspension roof support); they have a rectangular section and an arched or domed roof (Fig. 49).

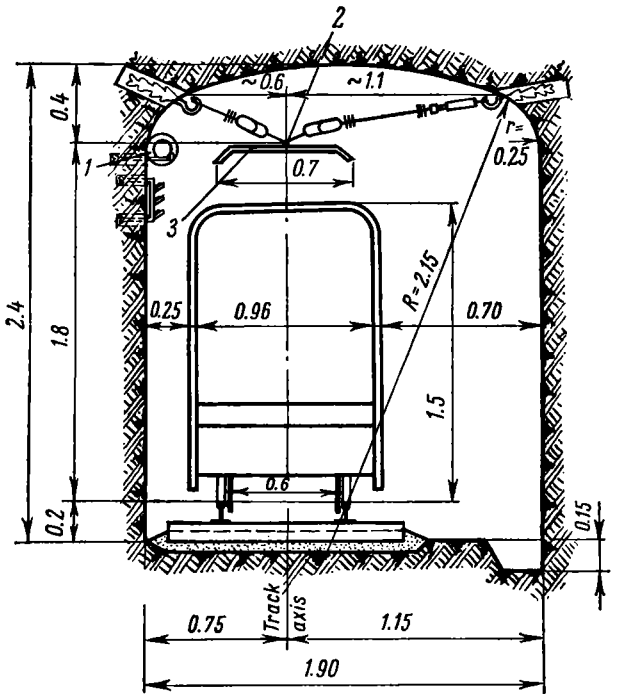


Fig. 50. Section of horizontal working for haulage with "Liliput" locomotives (untimbered):
1—compressed air pipe (d = 100 mm); *2*—trolley wire; *3*—current collector of locomotive

Single-track level mine openings with timber support have a rough section of up to 6 sq m when haulage is done by electric locomotives of the "Liliput" type, and up to 8-8.5 sq m with haulage by large-capacity mine cars and ordinary electric locomotives. The rough section of unsupported level mine openings is up to 4.5 sq m with the use of "Liliput" locomotives and up to 6-6.5 sq m with large-capacity mine cars and ordinary electric locomotives. Examples illustrating sections of level mine openings with haulage by "Liliput" locomotives are given in Figs. 50 and 51.

Table 8 lists data on the sections of production shafts as set down for designing Soviet non-ferrous mines.

Table 8

Production Shaft Sections

Annual	Mine output, thous. tons	Sectional shape	Hoisting	Support	Section, sq m	
					effective	rough
	30-70	Rectangular	Cage	Timber	7.74	11.2
		ditto	ditto	ditto	12.25	16.8
		ditto	ditto	Prefabricated reinforced concrete	12.25	18.5
		Round, d= 4 m	ditto	Reinforced-concrete tubbings	12.6	17.4
	100-300	Rectangular	Cage	Timber	18.6	24.2
		ditto	ditto	Prefabricated reinforced concrete	18.6	27.28
		Round	ditto	Prefabricated reinforced-concrete tubbings	23.7	30.2
		ditto	ditto	ditto	19.6	25.5
		Rectangular	Skip-and-cage	Reinforced concrete	18.6-23.1	27.3-33.0
		Round	ditto	Prefabricated reinforced-concrete tubbings	19.6-28.26	25.5-36.3
	300-600	Round	Skip-and-cage	Prefabricated reinforced-concrete tubbings	28.26-33.16	36.3-41.82
	600-1,500	Round	ditto	ditto	33.16-44.15	41.82-55.39

Bottom Stations of Production Shafts

Bottom stations of low-tonnage production shafts are ordinarily arranged on a spur-track pattern. With an adequate disposition and sufficient space of workings and with effective conduct of switching operations spur-track type bottom stations can also be used in mines with a fairly high output, up to 1-1.5 million tons a year. If the production exceeds 1.5 million tons per year sections of the circular-track type are set up.

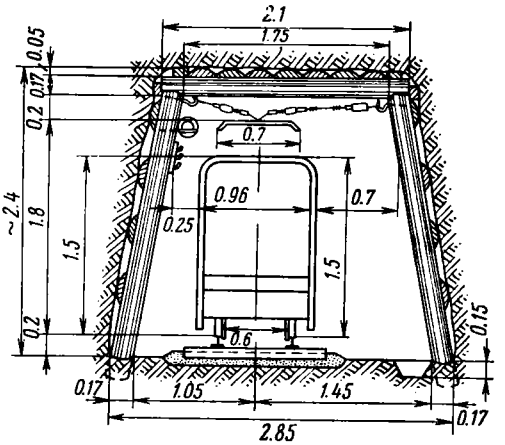


Fig. 51. Section of horizontal working for haulage with "Liliput" locomotives (timbered)

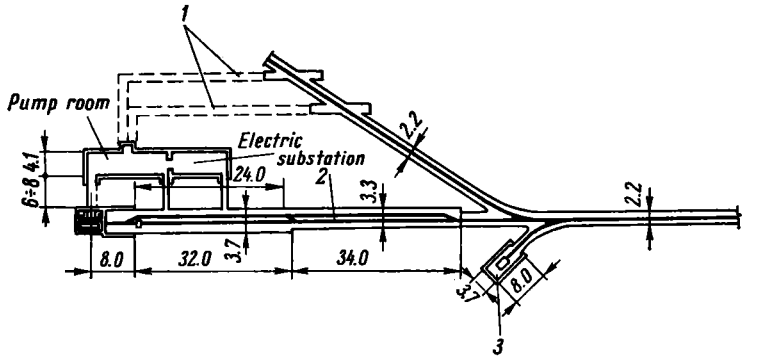


Fig. 52. Switchback-type shaft station of mine with annual output of 70 thous. tons:

1—storage sump; *2*—by-pass with crosscut length exceeding 100 m; *3*—underground shed for electric locomotives

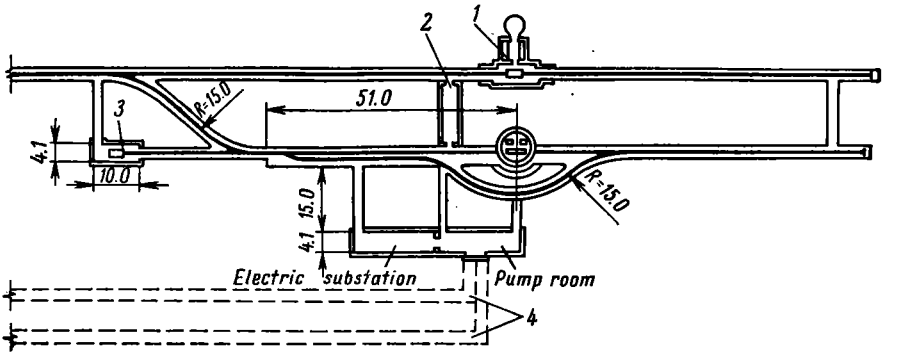


Fig. 53. Switchback-type shaft station of a mine with annual output up to 600 thous. tons:

1—discharging chamber; *2*—rest-and-waiting room; *3*—locomotive shed; *4*—main storage sumps

Table 9

Shaft Station Cubage as Stipulated in Projects Prepared by a Soviet Projecting Institute

Annual mine output, thous. tons	Number of shafts serviced by bottom station	Type of bottom station	Volume, cu m	Annual mine output, thous. tons	Number of shafts serviced by bottom station	Type of bottom station	Volume, cu m
30-70	1	Spur	450	600-1,500	1	Circular	3,930
100-300	1	Circular	1,390	600-1,500	2	ditto	4,625
100-600	1	Spur	1,367-2,945	600-1,500	2	ditto	6,225
600-1,500	1	ditto	2,945				

The cubage of a shaft station (cage-and-skip branches without adjacent rooms) varies from 450 to 6,225 cu m, depending upon the mine output and the number of shafts (Table 9).

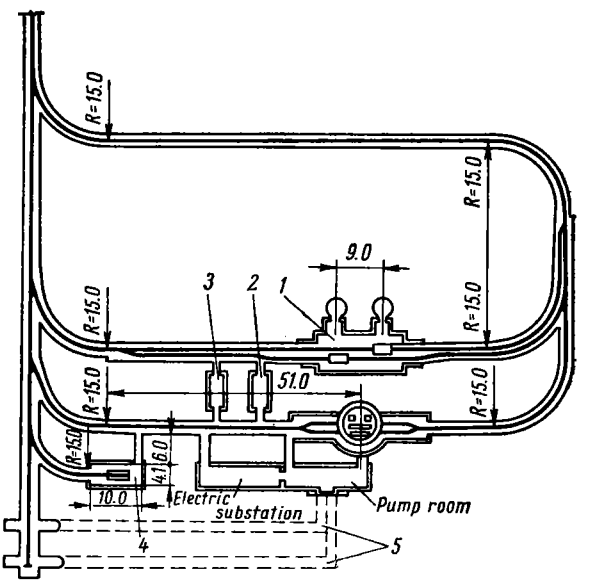


Fig. 54. Circular-type shaft station of a mine with annual output up to 1,500 thous. tons:
1—discharging chamber; *2*—rest-and-waiting room; *3*—underground first-aid room; *4*—locomotive shed; *5*—main storage sumps

The aggregate cubage of service rooms adjoining the bottom station (pump room, power substation, locomotive shed, measuring bins, etc.) makes up 50-100% of the shaft station volume.

Bottom stations in mines of different production capacity are shown in Figs. 52, 53 and 54.

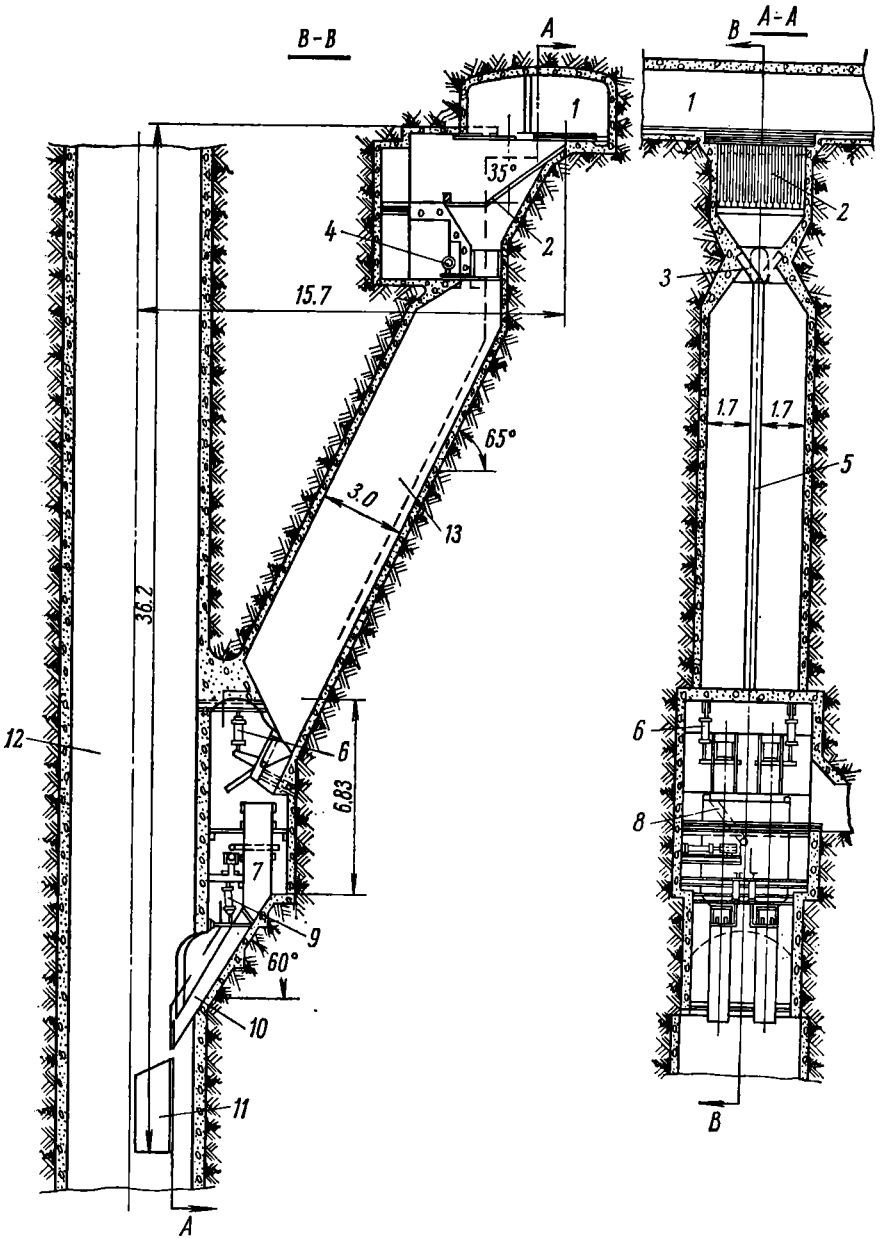


Fig. 55. Loading arrangement for skip hoisting with high-capacity pocket for ore:

1—skip branch of shaft station; *2*—bar grizzly; *3*—steel funnel-shaped chute with deflecting bottom gate; *4*—air-feed motor actuating deflecting bottom gate; *5*—reinforced concrete partition dividing pocket into two compartments; *6*—air-cylinder for opening and closing arc gate; *7*—measuring bin with capacity equal to that of skip; *8*—deflecting gate in upper portion of measuring bin for directing material into right or left compartment; *9*—air-cylinder for opening and closing measuring bin; *10*—chute; *11*—skip; *12*—shaft; *13*—pocket

The pump room, electric locomotive shed and other workings at a shaft station are excavated in the same fashion as in the opening of coal deposits. Bins, however, are usually designed for larger capacity (Fig. 55).

Auxiliary Shafts

Auxiliary or service shafts may be intended for various purposes, therefore their sectional area and number of compartments vary considerably. Service shafts are ordinarily supported by close or hanging timber sets throughout, and in cases of extensive service life they are lined with concrete or prefabricated reinforced-concrete support.

As for hoisting shafts, the section of ventilating shafts must meet ventilation requirements during exploitation of the mine, i.e. provide for the passage of the required volume of air at permissible velocity.

The above-mentioned Projecting Institute has adopted standard effective sections for ventilating shafts: rectangular 16.8-18.5 sq m, round 17.4-30.2 sq m.

3-5. GEOLOGICAL AND EXPLORATION BACKGROUND DOCUMENTATION AND COORDINATION OF GEOLOGICAL AND PROSPECTING OPENINGS WITH REQUIREMENTS FOR MINING ORE DEPOSITS

Before implementing this or that method of underground mining it is necessary to ascertain the possibility and expediency of open work, which has many substantial advantages over underground mining (see Chapter "Opencast Mining"). In order to choose the appropriate method of opening and solve other problems of working deposits one should first obtain exploration data needed for planning and projecting. These include: information on reserves approved by the Central Committee on Reserves (textual and graphic materials with estimation of ore and metal reserves in levels and blocks, along with characteristics of grades and classes of the useful mineral); a geological map of the region and occurrence to a scale of 1 : 10,000; lengthwise, transverse and horizontal sections to a scale of 1 : 500-1 : 1,000; sections of trenches, pits, boreholes with assay findings; a topographical map of the deposit to a scale of 1 : 1,000-1 : 2,000 with the outcrop of the mineral indicated and contour lines traced at every 2-5 m; topographic maps of the occurrence and of land plots adjoining it to a scale of 1 : 5,000; plans and sections covering all underground prospecting and other workings to a scale

of 1 : 500-1 : 1,000 coordinated with the surface relief and with assay findings in each individual working; a comprehensive report on the exploration work enclosing graphic material covering each individual working; data on the stability, hardness and water-bearing capacity of the solid mass of enclosing rocks and of the useful mineral, data on the humidity and "loosening" factor of the mineral; characteristics of the grounds and other information necessary for the selection of a site for the mine surface structures, plants and spur lines; hypsometric plans of the bottom and back in level and gently dipping deposits; hydrogeological characteristics of the occurrence with estimation of the expected water inflow; brief information on economic conditions attending exploitation of the deposit; meteorological data covering a number of years (temperature, atmospheric precipitations, winds).

Geological and exploration service workers must see to it that excavation of prospecting workings—shafts, adits, crosscuts, etc. is done properly. If prospecting workings are used for exploitation of the mine they must be made with due consideration for the requirements which such workings are to meet in respect of the site of sinking, the interval between levels and individual early development openings, their section, the permissible gradient and radius of the track curvature. Infringement of the rules established for making prospecting workings usually results in various complications and causes increased expenses and delays in carrying out exploration work.

More detailed information as to the coordination of prospecting workings with exploitation requirements is given below.

3-6. CHARACTERISTICS OF LOW-OUTPUT COMBINED EXPLORING AND PRODUCTION SHAFTS (50-200 THOUSAND TONS PER YEAR)

Section of the Shaft and of Horizontal Workings. The rough section of a single-cage hoisting shaft is 5.7-6 sq m and that of a double-cage shaft 10.0 and 10.6 sq m (depending upon the type of shaft timbering used).

The rough section of horizontal mine workings with haulage by "Liliput" locomotives is 6.0 sq m with support by timber sets (props 170 mm in diameter) and 4.5 sq m untimbered.

Horizontal mine workings are driven with a downward gradient of 0.003-0.005 in the direction of the hoisting shaft (equivalent resistance gradient for mine-cars with roller-bearings), the radius of the track curvature being at least 10 times as great as for the axle base with electric locomotive haulage (the axle base is the distance between car or locomotive wheel axles).

Hoist Performance During Exploitation is reflected in Table 10.

Table 10

Hoist Efficiency (tons/shift) in Relation to Hoisting Depth and Number of Cages Accommodating 0.35 cu m Mine-Car

Hoist	Hoisting depth, m			
	50	100	150	200
Single-cage	95	70	55	45
Double-cage	190	140	110	90

It is determined with an allowance for the time (30 minutes in each shift) spent in lowering and hoisting 45-110 men in a single-cage and 90-120 men in a double-cage hoist (depending upon the hoisting depth).

Hoisting Plant Employed in Mining. With single-cage hoisting—a БЛ 1,600/1,224 winder with a 35 kW, 725 rpm, 220/380 V electric motor.

With double-cage hoisting—2БЛ 2,000/830 winder with a 50 kW, 725 rpm, 220/380 V electric motor.

Cages of the standard type designed for ВОК-35 mine-cars; carrying capacity 3,500 kg, overall dimensions: length 1,300 mm, width 930 mm, height 2,700 mm, empty weight 1,350 kg. Accommodates 5 men.

Mine-cars of the standard ВОК-35 type with a hand-tipped side-dump body, capacity 0.35 cu m; track gauge 600 mm; load-weight of the car 600 kg, empty weight 400 kg; overall dimensions: length 1,180 mm, width 755 mm, height 1,092 mm. Hoisting ropes 18.5 mm in diameter weighing 1.2 kg/m; counterweight (with single-cage hoisting) 1,800 kg; pulleys 1.6 m in diameter.

Depending on the local topography headframes are made either 17 m high with the top landing platform arranged on a trestle disposed 7 metres above the shaft mouth or 10 metres high with the top landing platform arranged at the level of the shaft mouth.

Haulage in Mining Operations. The type of haulage depends upon the mine output and the required haul length.

With electric haulage, trolley locomotives of the "Liliput" 1ТЛ-1м type are employed. This locomotive has an adhesion weight of 3,100 kg for a 600 mm gauge track and a drawbar pull of 480 kg. Its average travel speed is 8.3 km/hr. It is capable of pulling a train of eight ВОК-35 mine-cars. Its overall dimensions are: length 2,712 mm, width 920 mm, height with current collector down 1,500 mm. The weight of 1 running metre of rails is 15 kg; movable-point switches are of the 1/4 model; use is made of baulk or plate sleepers 1,200 mm long and 110-120 mm thick.

Ventilation in Mining. Required volume of air 8-10 m^3/sec; low-pressure centrifugal fan No. 8 ensuring a throughput of 29,000 m^3/hr with maximum head of 40-45 mm water; 10 kW, 970 rpm, 220/380 V fan motor.

Drainage in Mining. Water inflow 30-50 m^3/hr; KCM-30 or KCM-50 pumps; pump motor power: with $Q = 30$ m^3/hr $N = 9$-36 kW, depending on suction head ($H = 50$-200 m); with $Q = 50$ m^3/hr $N = 14$-54 kW depending on suction head ($H = 50$-200 m); steel pipes 100 mm in diameter; size of pumping room for three pumps 15.5 × 4.0 × 2.5 m; storage sump capacity 8 hr of continuous water inflow.

Compressor Plant. In a mine with an output of 50 tons per shift compressed air consumption with allowance for leakages is 30 m^3/min. Compressor room for four 200B-10/8 compressors with a capacity of 10 m^3/min each; 80 kW, 720 rpm, 220/380 V compressor motor; weight of compressor 1,500 kg (the heaviest part weighs 380 kg); cooling water consumption 3 m^3/hr; two receivers of 8 cu m capacity each; seamless-steel compressed-air pipes. With mine output of 50 tons per shift and air pipeline length up to 1,000 m the major trunk line is made of pipes 100 mm in diameter, and distribution network of 50 and 36 mm pipes.

The above characteristics of combined exploring and production shafts are in keeping with commonly encountered mining conditions. Whenever the conditions differ from ordinary ones these characteristics should be revised.

II. METHODS FOR UNDERGROUND WORKING OF MINERAL DEPOSITS

General information on methods of working mineral deposits. By a method of working a mineral deposit (or a part thereof) is meant the aggregate of development and stoping workings and a definite order of their excavation, correlated in space and time. A properly chosen mining method should meet the requirements of safety, maximum economic efficiency, and high intensity of extraction. In the Soviet Union problems of safety are given special attention.

By safety is understood the provision of means for the preservation of life and health of the personnel and measures to secure the integrity and adequate condition of machinery, equipment and structures.

Economic efficiency of mining implies extraction of the mineral from the occurrence with the least possible expenditure of labour, power, and materials. It is especially important to ensure the maximum efficiency of labour, since wages comprise a considerable portion of mining expenses (40-50 per cent or more). In the mining industry of the Soviet Union optimal labour efficiency goes hand in hand with measures to lighten the labour of the miners. To this end use is made of machinery and of mining methods most expedient in a given set of conditions. Economic efficiency also implies the least possible losses and dilution of the mineral with barren rock during mining, the maximum possible extraction of valuable constituents and the lowest cost of their subsequent processing. The latter requirement is particularly important in mining non-ferrous and rare-metal deposits containing very valuable components, whose further processing at concentration plants and dressing mills requires application of very complex methods.

Efficient operation of these enterprises largely depends upon the quality of the ore processed. *In the final analysis economic efficiency in working mineral deposits implies maximal extraction of useful components with the least expenditure of labour and money at all stages of operation (mining, beneficiation, smelting).*

By intensity of mining is meant winning per unit time of a maximum possible amount of the mineral. The underground mining methods are classified as methods for working coal and placer, metalliferous and non-metalliferous deposits.

CHAPTER IV

METHODS OF MINING COAL AND PLACER DEPOSITS

4-1. BASIC FACTORS INFLUENCING THE CHOICE OF METHODS FOR MINING COAL DEPOSITS

Let us consider the principal factors that influence the choice of methods for mining coal occurrences. A number of features mentioned below are of importance in working ore deposits as well.

Shape and Thickness of a Deposit

In the ideal case, a mineral bed has a regular thickness. In coal mining practice, however, one frequently encounters considerable variations in thickness caused by disturbances of genetic and tectonic nature. An example of sheet deposits with a regular thickness is the Donets coal basin. Typical examples of deposits with a frequently changing mode of occurrence are coal beds in the eastern slope of the Urals mountain ridge. In selecting mining methods due account should be taken of frequent changes in thickness, roughness of the bottom and back, swells and petering out of beds.

In mining thin and medium-thick beds full-seam extraction is ordinarily practised, while in working thick beds either slicing or full-seam extraction is preferred. In thick beds the mined-out space often has to be stowed with mine-fill.

The minimum working thickness of a seam fluctuates within a wide range and depends upon the angle of dip, characteristics of the bed, machinery used and the nature of the reserves in a given basin. Thus, for instance, the minimum working thickness of seams in the Donets basin is set at about 0.5 m. The maximum working thickness of coal seams in different coal fields varies from 3 to 30 metres and more.

Dip Angle

An important factor in choosing mining methods is the angle of dip. With a heavy pitch, for example, the mineral moves by gravity and this is always taken into account in selecting the system of

mining. In coal mines, level, gently sloping (up to 25°), inclined (25-45°), and steeply pitching (45-90°) seams are distinguished.

Frequent changes in the dip angle within one and the same deposit tend to complicate its mining.

Barren Intercalations

Sometimes mineral beds contain barren intercalations. Their presence complicates mining operations, since the barren rock has to be thrown into the worked-out space, or used as mine-fill to avoid an increase in the ash content of the coal. To separate intercalations less than 5 cm in thickness is practically impossible and the gangue contained therein becomes mixed with coal. Considerable complications are caused by intercalations of barren rock susceptible to self-ignition. Rock from such intercalations should never be left in the mined-out space.

Properties of Coal

The basic properties of coal include rigidity, schistosity and cleavage, susceptibility to self-ignition, chemical composition and technological properties, size characteristics after breaking, firedamp evolution.

Rigidity of coal. By rigidity of coal is meant its resistance to mechanical effects. The rididity of coal is dependent upon its hardness, toughness, mineralogical composition and structure, presence of cracks and cleavage.

The rigidity of coal greatly influences the selection of undercutting and breaking methods. Classification of coal as to rigidity is done with special reference to technological operations, such as working with coal cutters, boring with electric rotary drills, etc. The rigidity of coal with regard to cutting with a coal cutter is, for example, characterized by the number of picks of a certain quality per 1 sq m of the cut.

Schistosity and Cleavage. Sedimentary rocks can be split along parallel planes of bedding. This property is termed *schistosity.* In some rocks schistosity is related to the time of rock formation (primary schistosity). Rocks often exhibit schistosity of secondary origin.

A system of joints of secondary origin, formed as a result of tectonic processes, which have transformed solid monolithic rock into schistose rock, is called *cleavage.* As distinct from primary schistosity, cleavage is generally non-parallel to the bed surface. The spatial position of cleavage planes is usually characterized by the angle between the cleavage orientation and the strike of the bed.

The cleavage planes and their spatial orientation strongly influence the choice of mining methods. Cleavage in the roof and its orientation in space are of great importance for roof stability. In the working of steeply dipping, and particularly of thick beds big lumps of coal may become detached along the cleavage surface. This hazard can be avoided by changing the sequence of coal extraction and using the appropriate support at the face.

Susceptibility of coal to self-ignition. This property of coal strongly affects the choice of mining methods. Seams of self-igniting coal should be extracted with the least possible losses and with close packing of the mined-out space, since with a rise in temperature the abandoned coal rapidly oxidizes and this often results in its spontaneous combustion. Fire hazards increase in mining thick beds. Extraction of coal containing minimum amounts of volatile matter (anthracite) presents less fire hazard.

Chemical composition and technological characteristics. The chemical composition and technological characteristics of coal are established on the basis of elementary and technical analyses. The former serves to determine the content of carbon, hydrogen, oxygen and nitrogen. The latter is of paramount commercial importance, for it indicates the content of moisture, ash, sulphur, phosphorus, the yield of volatile matter and the calorific value of the coal (this is largely dependent upon the content of combustible elements—carbon and hydrogen). In addition to the above-mentioned analyses coal is assayed for its ability for coking and preparation. Investigation of coal for its amenability to preparation is done with special reference to its grade size and fractions of different specific weight. The nature and technological characteristics of coal can best be defined by first analysing its petrographic properties, which is a special concern of geologists. In mining coal measures the grades of coal are taken into consideration when choosing the order of extraction and the methods for mining individual seams. The self-igniting properties of coal, which largely depend upon the content of sulphur, are taken into account in selecting mining systems and fire-fighting measures to be applied in the course of stoping.

Size of coal. The size and number of lumps in the extracted coal are of great importance for certain grades of coal (anthracite, gas coal). This should be reckoned with in selecting the methods of coal breaking and transportation.

Gas-bearing capacity of a deposit. Mining of coal beds may be attended by the evolution of various gases, such as methane (firedamp), carbon dioxide, carbon monoxide, hydrogen sulphide, hydrogen and gases released by blasts. Methane is released most frequently and in appreciable amounts. Methane evolution from coal and wall rocks takes various forms, such as placid outflow, relea-

se with audible sound effects (fumaroles), and sudden noisy outrushes with outbursts of fine coal. Sudden outrushes are the most dangerous, since vast amounts of gas and coal (from several tons to several hundreds and even thousands of tons) are ejected.

Gassy mines are classified in accordance with the amount of released methane or hydrogen, as indicated in Table 11.

Table 11

Categories of Mines According to Abundance of Gas

Characteristics	Categories of mines			
	I	II	III	Supercategory
Amount of methane and hydrogen released within 24 hours per ton of average daily output, cu m	up to 5	5-10	10-15	Over 15
Minimum amount of required air per ton of average daily output, m^3/min	1.0	1.25	1.5	Based on 0.75% dilution of gases in total return current

Note: 1. In classifying mines according to gas abundance 1 cu m of hydrogen should be taken as equivalent to 2 cu m of methane.
2. Mines with seams hazardous in respect of outbursts and fumaroles are included in the supercategory group.

Evolution of methane, as a rule, increases with depth. This is confirmed by the data for the Donets basin:

Depth of mine, m	1-150	150-250	250-350	350-450	450-550	550-800
Amount of methane given off within 24 hr per ton of coal output, cu m	1.2	5.7	9.5	11.3	16.3	20.0

Sudden outrushes begin at a mining depth of 200-300 m. They occur in many coal fields and regions.

The distribution of gas in coal measures, including the formation of gas foci, is in the main determined by geological factors. It is primarily the geological structure of the deposit and tectonic disturbances that determine the nature of coal and gas outbursts. Their foci are mostly in areas and zones of plicative dislocations of coal and rocks (flexures, squeezes and bulges). In disjunctive zones (involving discontinuity of rocks) almost no sudden outrushes of coal or gases take place. Hence in mines where this phenomenon occurs it is necessary to carry out a detailed study of all disturbances with their subsequent classification. Investigations conducted by Soviet geologists for many years have also established that hydrogeological conditions determine the gas-bearing capacity of the entire coal stratum and consequently sudden outbursts of coal and gas.

Damp beds have only a small gas-bearing capacity and usually no outrushes occur in them. In studying such beds due account should be taken of the general trends in the distribution of gas in the coal stratum, of rock pressure and physico-mechanical properties of the rocks, i.e. the environmental phenomena. Of great practical significance is drawing up (in cooperation with exploitation personnel and geologists) of maps showing the gas abundance of coal measures.

The discharge of methane is one of the prime factors to be considered in choosing underground mining methods. Being lighter than air, methane tends to fill the space of ascending mine workings. One of the major means to reduce the concentration of methane is the application of mining methods involving a minimum of dead faces and upraises.

In mines presenting hazards of sudden gas discharges with coal outbursts the following additional steps should be taken: drilling of advance exploration boreholes at the faces of development and prospecting openings; application of mining methods involving a minimum of development work and using production stopes of regular shape, without projections; beginning the mining with such protective beds in the formation as are not liable to produce sudden outrushes of coal and gas; degasification of a seam by boring large holes below the production level; the use of shock blasting, i.e. multiple-hole blasting of heavier charges than in ordinary breaking of coal; application of rotary drilling and blasting when breaking coal in development headings and stopes without the utilization of percussion tools and machines; realization of other safety measures, such as blasting between shifts in the absence of miners, sectional ventilation scheme, setting up shelters, providing all workers with personal rescue equipment, etc.

It should be borne in mind that in mining coal deposits there may be sudden releases not only of methane, but also of carbon dioxide.

Properties of Wall Rocks

In the conditions of mining coal occurrences the wall rocks are commonly made up of clayey, sandy and coaly shales, sand- and limestones. A knowledge of wall rock properties is needed to enable one to solve problems linked with the excavation and timbering of mine workings, rock pressure control and the application of the appropriate mining system.

Strength of rocks. By the strength of rocks is meant the ability of a solid body to withstand the action of definite forces producing a particular deformation. It is characterized by ultimate strength

of the rocks (in kg/cm^2) in tension, compression, bending and shearing. These are determined by laboratory tests of rock specimens. The strength characteristics of rocks depend upon the size of the grains making up the rock, the binding agent, porosity, water-content, and also the size, shape and stress condition of the specimens. Coarse-grained and porous rocks are softer than fine-grained ones. It should also be noted that findings of rock specimen tests cannot be applied to all rocks of the same petrographic composition, and therefore tests should be conducted for each individual deposit. Much attention should be paid to establishing the strength of rocks, for this determines their stability, which is of prime importance in mining.

Stability of rock exposures. The deformation of a rock exposure may for a long time remain within the elastic limits. In such instances the exposure is regarded as stable and no timbering is normally needed. Should the deformation of a rock exposure surpass the elasticity limit the exposed rock surface changes its position. In this case the state of the exposure is designated as unstable. Deformation of such exposures often ends in sliding or caving of individual sections and pieces of rock. Exposures of friable rocks liable to collapse without any discernable deformation are also unstable.

In assessing the stability of rocks one should consider their strength and, in particular, the presence of jointing, both of primary and secondary origin.

The stability of wall rock exposures in the roof and floor is one of the major factors to be considered in selecting the system of mining, rock pressure control and design of support to be used in the stoping area. In prospecting and pilot exploitation of a coal deposit the properties of the rocks in the roof directly adjoining the seam (immediate roof) and of those lying at some distance therefrom (main roof) should be established.

In drawing up full characteristics of wall rock properties account must be taken of the joint effect of such factors as the depth of occurrence, the presence or absence of water, the stratigraphic nature of the rock sequence, the thermal conditions and tectonics of the region. It should be borne in mind that all kinds of tectonic disturbances with and without discontinuity of the rock generally lower its strength.

Mutual Arrangement of Seams in a Series

In mining closely spaced beds a definite sequence of extraction and a more-complex-than-usual support of the mined-out space should be adopted. These two factors require the application of strictly specified mining methods.

Water-Bearing Capacity of a Deposit

When exploring a deposit its water-bearing capacity should be ascertained. In the case of water abundance means should be provided for the drainage of individual sections, protective pillars left near water bodies which cannot be drained for some reason, a safe method of roof control adopted, etc.

The Effect of Mechanization on Mining

Mechanization of mining operations requires adequate conditions ensuring its efficiency. Therefore there is a substantial difference in mining methods practised with extraction by hand and with the use of mechanical equipment, even under similar mining and geological conditions. For example, the employment of coal cutters in production faces for coal undercutting and of conveyers to transport coal along the stope makes it necessary to adopt longer faces than is habitual with manual breaking of coal.

The use of cutting machines and coal miners in steep seams has necessitated substitution of the continuous for the overhand type of stope. Hand tramming along intermediate entries has been superseded by mechanical haulage and this has led to a sizable increase in the distance between gravity inclines. When planning the mining of a deposit one should choose a method best adaptable to the operation of the available machinery; the latter, in turn, should be selected so as to fully conform to the conditions of mining.

The cited list of major factors influencing the choice of a mining method proves the complexity of the problem.

The choice of a proper mining method greatly depends upon the initial data, i.e. the mining and geological characteristics of the deposit furnished by the staff of the geological and prospecting service. The completeness and correctness of these characteristics (the basic factors pertaining to the mining and geological characteristics are enumerated above) determine the quality of a project for working this or that deposit and its practical implementation. In preparing mining and geological characteristics of a deposit not only the useful mineral and the rocks directly adjoining the mineral body should be considered, but also the rock mass in the foot and, especially, the hanging wall, since the stability and other properties of the host rocks strongly affect the choice of the mining system and of the method of rock pressure control. Particular attention should be given to the water and gas abundance in the deposits, for should it exceed the normal values, special measures will be required to ensure proper and safe working of the deposit.

4-2. CLASSIFICATION AND CHOICE OF MINING METHODS IN WORKING SHEET DEPOSITS

So far there exists no unified classification of methods for mining sheet deposits in the USSR.

In the majority of proposed classifications of methods for underground mining of coal deposits (those of Prof. B. Boky, Academician L. Shevyakov and others) adopted in the Soviet Union the order and interrelation of making productive and development workings are accepted as the basis for classification.

In grouping up the classes of mining methods the characteristic feature is the general direction of movement (advance) of the stoping front (within a level or a panel) with reference to the elements of the occurrence of the bed, i.e. along the strike, on the rise, on the dip and obliquely to the strike. Low and medium-high seams are extracted throughout at a time, while high seams are worked throughout or are divided into separate slices, each being mined as a medium-high seam, and all slices as a coal measure. This points to the expediency of classifying the mining methods in accordance with the thickness of the bed (thick, thin and medium-thick beds).

Given below is a somewhat abridged classification of methods for underground mining of coal deposits.

This classification is very similar to that proposed by Academician L. Shevyakov.

A. Methods of mining low and medium-high seams

I. Continuous (longwall) mining.
II. Pillar mining:
  (1) long-pillar method;
  (2) pillar-and-stall method.[1]
III. Methods of mining by strips (layer-wise).[1]
IV. Room mining.[1]
V. Combined methods:
  (1) room-and-pillar system;
  (2) continuous and pillar (twin entry) method.

B. Methods of mining high seams

I. Methods of mining thick beds without subdividing them into slices:
  (1) long-pillar methods;
  (2) pillar-and-stall method;[1]
  (3) mining by strips;
  (4) room mining;
  (5) combined mining.

[1] Rarely used.

II. Methods of mining thick beds, dividing them into slices (slice mining or slicing):
(1) inclined slicing;
(2) horizontal slicing;
(3) transversely inclined slicing; [1]
(4) diagonal slicing; [1]
(5) vertical slicing. [1]

The above classification does not indicate groups of mining systems (direction of extraction) and the method of rock pressure control, such as caving of enclosing country rocks, filling of the mined-out space, gradual subsidence of host rocks. These features influence the general design characteristics of the mining system and its technical and economic characteristics.

The said features should be reflected in the full designation of the particular mining system; for example, long pillar mining along the strike with roof caving.

When discussing individual mining systems in the subsequent text we shall cite full designations of the methods under study. Rarely used methods are not considered in the book.

Basic considerations in choosing a coal-mining method. The selected method should comply with the above-mentioned basic requirements. When conditions permit several systems of mining (ordinarily not more than two or three) to be employed and these meet safety requirements the final choice is made by comparing technical and economic characteristics of stoping. The method ultimately adopted should ensure maximum extraction of the mineral from the occurrence at the lowest cost, with the quality of the mineral satisfying stipulated requirements.

4-3. ROCK PRESSURE CONTROL

Basic Concepts and Development of Theoretical Considerations

Rock pressure. Prof. V. Slesarev defines rock pressure as a force set up by the movement of rocks and their deformation caused by the disturbance of the equilibrium of these rocks as a result of the excavation of mine workings.

In this connection V. Slesarev emphasizes that generally rock pressure is not a permanent, invariable phenomenon; on the contrary, it changes with time.

This statement finds practical confirmation, in that one and the same mine opening exhibits "virgin" (primary) and "balanced" rock pressure. Prof. M. Protodyakonov wrote: "The pressure on the tim-

[1] Rarely used.

bering does not develop all of a sudden. Once the dislocation of rock has started the pressure begins gradually to rise, reaching after some time a certain maximal, sometimes quite appreciable, value finally breaking the installed timbering."

Of the latest, more clearcut definitions of rock pressure those made by A. Borisov, Professor of the Leningrad Mining Institute are worthy of note: "In a narrow sense, by rock pressure are meant loads bearing on natural and artificial supports (timbering, pillars, etc.)... In a broad sense, by rock pressure is meant a set of processes caused by the excavation of a mine opening (changes in the stressed state of the rock, breakdown of rock and timbering, etc.)."

Rock pressure control. Rock pressure control comprises an aggregate of measures aimed at artificial regulation of the rock pressure in an underground mine opening to secure safety and high efficiency of operations. Rational rock pressure control is of prime practical importance in working bedded deposits.

P. Tsimbarevich notes that all active factors which are of significance in rock pressure control may be classified as permanent and variable ones. The permanent factors include: natural factors—the geological and hydrogeological conditions, the thickness of the bed and its angle of dip, the physical and mechanical properties of the rocks and of the useful mineral; operational factors—the method of mining, the vicinity of other seams, worked out or being worked, etc. Under the heading of variable factors come: the method adopted for supporting the mined-out space, the size of the active stope area, the rate of the face advance, the methods employed for supporting mine workings, the procedures for extracting the mineral, the orientation of the stope with respect to structural elements of wall rocks, the shape of the stope face, etc.

The problem of rock pressure control may be formulated as finding and applying an optimal combination of values for variable factors that become valid in the presence of preset permanent factors. This problem is solved essentially on the basis of experience, and the mine timbering plays a particularly important role here as one of the variable factors in rock pressure control.

Methods of Studying Rock Pressure and Its Manifestations

In solving problems of rock pressure use is made of field experiments and laboratory and analytical methods.

Field experiments, i.e. observations and tests made in mines and on the surface, furnish a great deal of information useful for the practice of rock pressure control and for elaborating new hypotheses and laws. The potentialities of these procedures are, however, limi-

ted by difficulties encountered in making observations and generalization of their results. The main objects of rock pressure studies are: dislocation of wall rocks recorded with the aid of special observations carried out by mine surveyers; loads borne by the timbering determined by means of dynamometers of various design; areas of stable roof and floor exposures; special boreholes for estimating stresses in the rock masses (pressure relief method).

Laboratory methods are subdivided into two main groups: methods for investigating physico-mechanical properties and methods for simulating processes taking place in mine workings, such as the method of simple models, centrifugal modelling, modelling with the use of equivalent materials, and the optical method.

Method of simple models. Investigations are usually done on pieces of rock or on other materials of certain shape with holes imitating underground workings with and without timbering. Loads are applied on the perimeter of the model by a press or special weights. Due to the difficulty of observing the laws of similarity the tests yield but an approximate idea of the manifestations of rock pressure.

Centrifugal modelling is based on replacing the natural volumetric gravitational force by centrifugal forces of inertia applied to the model. Since laboratory models have small linear dimensions they are placed into a centrifuge the rotational speed of which is determined in advance with allowance made for the scale of the model, so as to bring the model to a stressed state corresponding to that existing in natural conditions. Centrifugal modelling allows volumetric problems of rock pressure to be solved, simultaneously observing geometric and dynamic similarity. Among the shortcomings of this method are the small size of the models, the difficulties of their manufacture from solid rocks, particularly from laminated ones, the difficulty of reproducing mine openings in the model and restricted possibilities of visual observations.

Modelling with the use of equivalent materials is based on the utilization of artificial materials, which, according to the laws of mechanical similarity, should reproduce in the model deformations, dislocations and failure of rocks occurring under natural conditions. Scientific substantiation of this method was furnished by G. Kuznetsov, a Soviet scientist. Today this method is widely applied in studying many aspects of rock pressure. Models geometrically similar to natural conditions are commonly made to a scale from 1/200 up to 1/20. The equivalent materials employed are powdered multicomponent mixtures of solid minerals (the basic part being quartz sand) cemented with paraffin, and gypsum-concrete compositions of different proportions. When making models (plane and three-dimensional) equivalent materials are placed in layers on special stands. Holes resembling mine openings are previously

left in the materials. Observations during extraction of a part of the model (an analogy of excavation) are made visually with the aid of instruments and apparatuses, such as tensometers, microdynamometers, etc.

The optical method of investigating stresses and deformations is based on the property of certain transparent isotropic materials (celluloid, gelatin, etc.) to assume under load an ability for temporary double refraction of polarized light rays. When passing through a stressed optically active model the polarized ray splits into two rays with oscillations in the planes of action of the two main normal stresses. When the oscillation planes of both light waves are brought into coincidence again with the aid of an analyser a series of bands of different colour and intensity—so-called isochromes—show on the screen. The colour and brightness of the isochromes indicate the distribution of the main tangential stresses in the model. The optical method plays an ancillary part, it aids in determining the qualitative picture of stress distribution and quantitative trends towards variation of stresses under the effect of various factors.

Analytical methods. The purpose of these investigations is to set up a mathematical expression for functional dependence of rock stresses and deformations around underground mine openings, on the one hand, and basic mining, geological and operational factors, on the other. In analytical methods of investigation use is made of theoretical propositions concerning the elasticity, plasticity and mechanics of loose bodies, or of rock pressure hypotheses, for example M. Protodyakonov's hypothesis about the dome or arch shape in designing support of horizontal (lateral) openings, the hypothesis of uniform stress distribution in a pillar cross-section used in designing interchamber pillars according to L. Shevyakov formulas, etc.

Hypotheses on Phenomena Occurring in the Roof of a Continuous Face with an Easy or Low Dip

In continuous production faces with an easy or low pitch the wall rock pressure control largely amounts to controlling the pressure exerted by the rocks making up the roof of the bed, i.e. to roof control. Many aspects of this problem are still obscure, and therefore in practice relevant hypotheses are used (Fig. 56).

Dome hypothesis. This hypothesis assumes the formation, in the initial stage of stoping, of a dome with natural equilibrium of the roof rocks over a mine opening (according to M. Protodyakonov). The rock pressure upon the timbering of a mine working cannot be greater than the weight of the rocks lying beneath the dome. With strong rock in the roof the pressure may be reduced to zero. As the span width of the mined-out area increases the height of the possible

domes of natural equilibrium rises too and so does the pressure on the production face timber. Later on, as the stoping operations progress, the nature of pressure on support in the worked-out space will depend upon whether the adopted practice consists in stowing or caving of the enclosed rock.

Back slab or cantilever hypothesis. In caving of the roof the back slab hypothesis can be applied. This assumes that the rock over the work area is held in place largely by the forces binding it with the rock remaining intact.

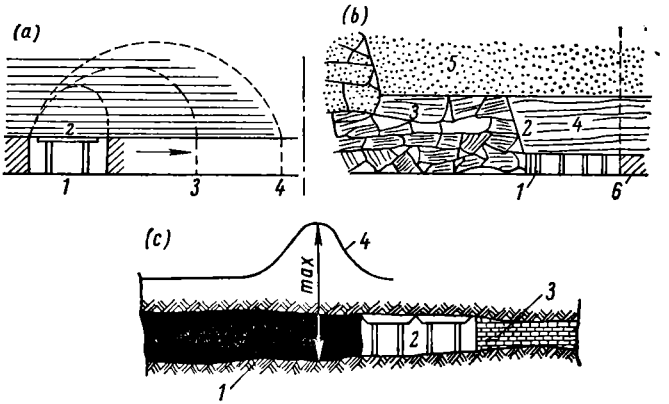


Fig. 56. Diagrams illustrating hypotheses concerning phenomena taking place in the roof of a continuous face (sections across the strike of the seam):
a—dome hypothesis (at start of mining in continuous production face): *1*—holding chute; *2*, *3*, *4*—natural equilibrium domes developing with progress of stoping); *b*—back-slab hypothesis (in caving of roof rocks): *1*—organ timbering; *2*—break or rib line of roof; *3*—caved rock of immediate roof; *4*—back slab; *5*—main roof; *6*—extracted seam; *c*—pressure wave hypothesis (with packed stope area): *1*—seam; *2*—timbered fore-breast area; *3*—mine fill; *4*—pressure wave (moves with face advance)

The minimum rock pressure is observed directly at the face (near the base of the back slab), where light timbering is sufficient to withstand the pressure. As the distance from the stope breast (from the back slab base) increases it becomes necessary to put up a special reinforced mine support. Further away from the face, in the direction of the mined-out space, the reinforced support will be insufficient and subject to deformations, while the rock will continue subsiding. Moreover, the rock lying directly over the work area will start coming down too under the effect of the forces of mutual cohesion.

In caving, roof control is effected by allowing periodical caving of the back as the length of the back slab increases, at the same time maintaining artificial support in the stoping area. To make the roof cave in, the timbering is knocked out, sometimes blasting is used. Ordinarily caving is followed by a substantial reduction

in the pressure upon the production face until a fresh back slab has been formed. The length of the back slab depends upon the properties of the rock, therefore the distance between consecutive induced cavings of the roof (caving space interval, or run) is subject to variations.

It is easiest to control the caving of clay and sandy shales, while in the case of sand and limestone shales roof control often entails considerable complications. Firm shales and sandstones lend themselves to systematic destruction only when broken by joints. The caving interval is determined by experience. It varies from 1.5 to 10 m (1.5 m with a weak, loose roof). In mining a low seam its back may come down to the bottom prior to actual caving, especially in the case of a heaving floor. Then no artificial caving is needed.

Pressure wave hypothesis. This is applicable when the worked-out space is supported by mine filling. It assumes subsidence of the roof rocks in the stoping area without any discontinuities, or with insignificant discontinuities. The caving rock exerts a strong pressure, which is sustained by the mine-fill; it becomes compacted, thus ensuring a gradual subsidence of the rocks. At some distance from the face there is a section in the solid mass of coal which is a sort of a fulcrum for the subsiding rocks. Under the action of evenly subsiding rocks the coal seam undergoes different stresses which gradually shift as the production face advances, i.e. a so-called pressure wave is observed. In this case, roof control consists in regulating the degree and character of the pressure upon the face which can be utilized for slackening the solid mass of coal. This is achieved by selecting the most suitable distance between the production face and the fill and the proper packing of the latter.

A pressure wave also forms in working without mine-fill, when the back and bottom converge, for example in swelling and upheaving of the floor.

In addition to the above-discussed hypotheses, which have found the widest recognition, hypotheses of articulated blocks, plastic extrusion of rock from the zone of bearing pressure, step-wise roof subsidence, etc. have been proposed.

The hypotheses considered allow one to judge the phenomena occurring in the roof of a continuous production face merely in a general way and only serve as a guide in choosing a method of rock pressure control. To make a more judicious choice of the rock pressure control method it is necessary to know the regularities in the manifestations of rock pressure under different mining conditions, this being within the domain of soil mechanics. The present stage of soil mechanics in the USSR enables solving a number of important problems concerning rock pressure control, such as estimation of the pillar size with different mining methods, determination of the size of mine

timbering, setting down conditions for the operation of mining machines, etc. However, regularities in the manifestations of rock pressure have not been established as yet because of the lack of conclusive information on the physical and mechanical properties of the whole complex of overlying and underlying rocks.

At present, the choice of a roof control method is based on an approximate theory of rock pressure with due consideration for facts established by the practice of mining mineral (especially coal) deposits. In order to ascertain the regularities in rock pressure one has to solve a whole series of complex problems of soil mechanics, the first task being to clarify the physical and mechanical properties of rocks with special reference to actual conditions of their deformation in the course of mining operations.[1]

The above considerations and properties of rocks should be taken into account in selecting a method of rock pressure control.

Selection of a Method of Roof Control

One of the major properties of roof rocks that should be considered in choosing the mode of roof control and, consequently, of support in the mined-out space is their cavability.

It is necessary to distinguish the immediate and the main roof, primary and secondary rock pressure. By the immediate roof are meant rock layers lying over the mineral bed. It may consist of several individual beds or bands (benches).

The immediate roof generally cannot remain within the boundaries of the worked-out space for long without timbering, and it easily caves in after removal of the timbering.

The main roof represents rock layers lying over the immediate roof. Sometimes, in the absence of an immediate roof the main roof lies directly over the bed. The pressure exerted by the main roof rocks is sustained by the coal pillars and packing, or by the caved-in rock. The main roof may be exposed over a large area. It collapses only after a considerable lapse of time.

Apart from the immediate and the main roofs there exists a false roof, which is a readily collapsible tender roof of small thickness (from 0.3 to 0.4 m) lying directly above the bed. This roof is capable of easily caving in together with the coal during its extraction. Primary rock pressure is due to the pressure of the immediate roof.

[1] For details on aims and methods of studying mechanical properties of rocks see: P. Tsimbarevich, *Soil Mechanics*, Ugletekhizdat, 1948; G. Kuznetsov, *Mechanical Properties of Rocks*, Ugletekhizdat, 1948; L. Baron *et al*, *Determination of Rock Properties*, Gosgortekhizdat, 1962; M. Protodyakonov, *Hardness and Drillability of Rocks. Collected articles, Problems of Mining*, Ugletekhizdat, 1958; K. Ruppeneit, *Mechanical Properties of Rocks*, Ugletekhizdat, 1956.

This pressure ordinarily manifests itself within the stoping area. Primary settlement, or subsidence of rocks is associated with primary pressure.

Secondary pressure owes its origin to the pressure exerted by the main roof. Secondary pressure is responsible for secondary subsidence of rocks.

The degree of rock cavability depends on: the petrographic composition (sandstones and limestones have a lesser tendency towards caving, while shales and schists collapse readily); the structure (lamination and jointing of roof rocks contribute to their collapse); the mechanical properties (compressive, tensile, bending strength and resistance to a more complex combination of deformations); the petrographic and mechanical properties of the rocks, which largely determine the nature of roof caving and its periodicity (caving space interval); the humidity (under a prolonged action of water rocks become less resilient and more prone to caving); time (the rate of the face advance and the size of the overhanging back slab of the roof are directly related to one another); the susceptibility of the bottom to upheaving.

All these factors should be taken into account in prospecting and exploitation of mineral occurrences.

In the USSR a great deal of work on classification of roof rocks with a view to the proper choice of rock pressure control has been done by research institutes of the coal industry. In the Donets and other coal fields the classification of roof rocks proposed by one of the Soviet planning institutes of the coal industry has become very popular for selecting the mode of rock pressure control in mining slightly inclined and sloping seams. According to this classification the roof rocks are divided into four classes with the following methods of roof control (Fig. 57).

Class I. The immediate roof consists of an easily collapsible rock. The thickness of the rocks in the immediate roof is not less than a 6- to 8-fold thickness of the minable coal seam. The recommended method of roof control is complete caving. As the immediate roof caves in the rock fills the entire worked-out space and protects the main roof from collapsing (Fig. 57*a*).

Class II. The immediate roof contains a mass of readily caving rocks whose thickness is less than a 6- to 8-fold thickness of the minable coal seam. The main roof includes a difficultly caving mass of rocks, which collapses only some time after the production face has been pushed forward and a considerable area of the roof exposed.

The suggested mode of roof control is partial caving between constructed rock packs (Fig. 57*b*). The rock packs are 4-6 m wide and spaced at a distance of 8-20 m. The rock for the construction of packs may be obtained by ripping the roof of so-called dummy

roadways or gathered in caved-in sections of the mine. The latter method is used relatively seldom. With ripping of the roof in dummy roadways these are commonly made 2-4 m wide, the thickness of the rock ripped being 0.5-1.5 m. Near the breast of the face dummy roadways are provided with support. The timbering of dummy roadways is moved forward as the production stope is advanced.

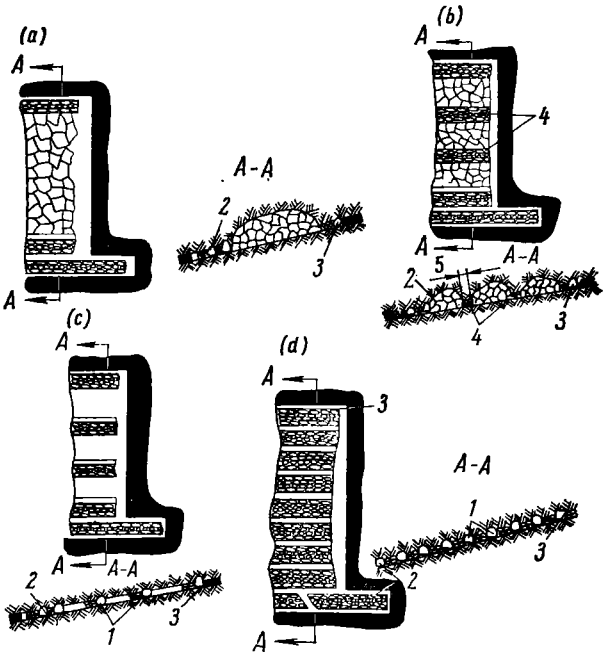


Fig. 57. Seam extraction with different methods of roof control (timbering of workings and of face breast not shown):

a—roof control by complete caving; *b*—roof control by partial caving (roof is caved between strip or rock packs); *c*—roof control by partial stowing with mine-fill obtained by ripping back of dummy roadways; *d*—roof control by complete packing with mine-fill obtained by ripping back of dummy roadways; *1*—dummy roadways; *2*—haulageway; *3*—airway; *4*—rock packs

Class III. The immediate roof is comprised of a more or less thick stratum of difficultly caving rock, or there is no immediate roof at all and the working seam is overlain by the main roof, which may be exposed over a considerable area.

The recommended method of roof control is partial stowing. The width of packs and their spacing are the same as in partial caving. The space between rock packs is not caved (Fig. 57*c*).

Class IV. The roof consists of rocks capable of gradual and uniform subsidence by 0.8-1 m without developing any ruptures or fissures.

The suggested mode of roof control is smooth and uniform subsidence. This method can be applied with seams of limited thickness.

Complete stowing is employed when none of the above-mentioned methods yields satisfactory results (with a very loose roof, with a fast but extremely fractured roof, in extracting self-igniting coal, in mining closely spaced seams), and also when it is necessary to protect surface structures and plants from ground movement. Complete stowing of the worked-out space may be done by packing it with the rock obtained by ripping the roof of dummy roadways driven at a small distance from one another (Fig. 57*d*), or with rock supplied from outside. In stowing with the rock ripped from the roof of dummy roadways the latter are not packed. When using rock supplied from outside the mined-out space is packed with the aid of stowing machines, or filled by flushing. The latter method is used in mining steep seams.

A more detailed characteristic of enclosing rocks is given in classifications prepared with special reference to individual coal fields (Donets, Kuznetsk and others). Complete caving is the least labour-consuming method of roof control. It is suitable for many coal basins and should be considered the basic mode of roof control. Today coal output from walls with roof control by complete caving comprises 90% of the entire underground coal production in the USSR.

Support of Production Faces and Caving of the Roof

Combination of certain types of support with correct methods of roof control is of major practical importance and ensures successful stoping. The support of production faces is also of utmost importance for the safety of mining, and hence this problem should be given much attention.

The use of wooden timbering. At present wooden timbering finds only limited use in Soviet coal mines. Below we give brief information on the subject.

Face timbering is employed with different modes of roof control. To construct it, parallel rows of timber props 15-20 cm in diameter are set up with the space between the rows ensuring the proper disposition of conveyers and flitting of coal mining combines or cutters (800-900 mm). The distance between the props in the row is set at 0.7-1.1 m. The props are made of round softwood. The diameter of props may be calculated by using M. Protodyakonov's formula

$$d = \sqrt[3]{\frac{Pl}{5.35K}}$$

where d = prop diameter, cm;
P = breaking load (20,000 kg);
l = prop length, cm;
K = strength of pinewood comprising 270 kg/cm².

Considering the values of P and K, $d = 2.4\sqrt[3]{l}$.

The diameter of the prop can be determined more accurately by calculation for longitudinal flexural strength. [1]

The props are wedged under head boards. These may be made of wood slabs, lagging slabs or sawed lumber. The head boards may

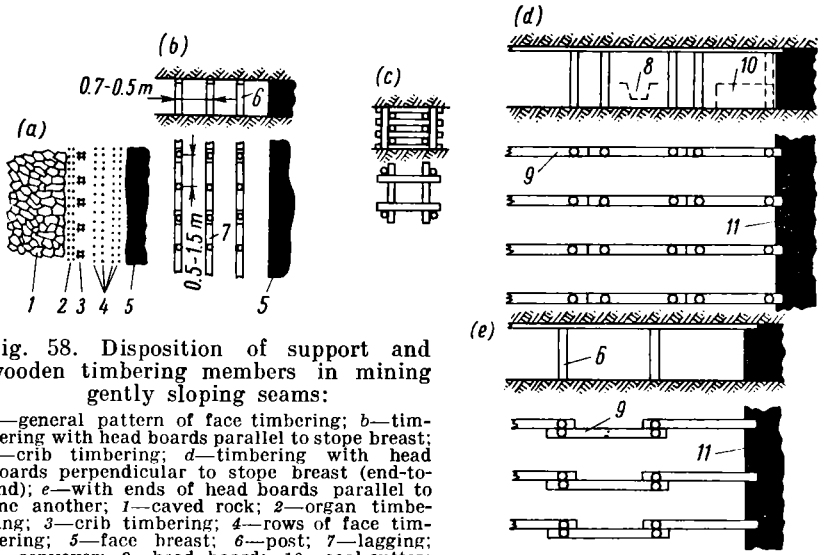


Fig. 58. Disposition of support and wooden timbering members in mining gently sloping seams:

a—general pattern of face timbering; *b*—timbering with head boards parallel to stope breast; *c*—crib timbering; *d*—timbering with head boards perpendicular to stope breast (end-to-end); *e*—with ends of head boards parallel to one another; *1*—caved rock; *2*—organ timbering; *3*—crib timbering; *4*—rows of face timbering; *5*—face breast; *6*—post; *7*—lagging; *8*—conveyer; *9*—head board; *10*—coal-cutter; *11*—solid coal in place

be placed parallel or perpendicular to the stope line, overlapping or abutting each other. When set up perpendicular to the stope line one end of the head boards is brought into a hitch made in the roof of the bed. After a cutting machine has made the first cut in the coal timber posts are set up directly at the face (Fig. 58). When necessary, the roof is lagged with slabs of timber in addition to the head boards.

In caving and sometimes when the roof subsides smoothly and uniformly *special support* is employed in addition to that put up at the face. Special support includes crib, organ and cog timbering. A crib made up of props laid to form a square is usually put up near

[1] See P. Tsimbarevich, *A Course of Mine Support*; V. Slesarev, *A Course of Mine Support.*

the posts of face timbering, and is thoroughly wedged for stability. Cribs are set up in single or double rows along the working face. In each row the cribs are ordinarily spaced at a distance of 1-2 m and more (Fig. 58).

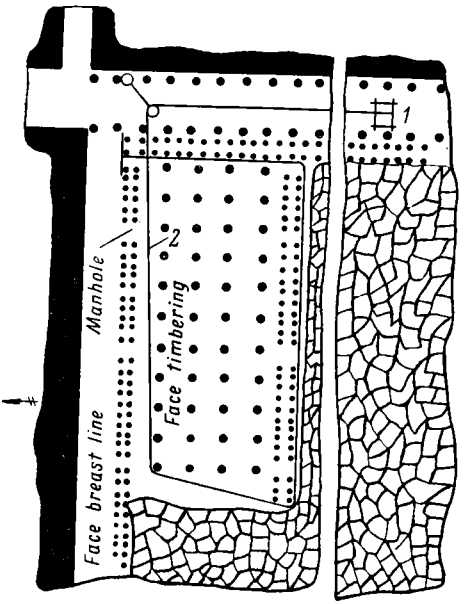


Fig. 59. Mechanical prop pulling arrangement:
1—prop-pulling tugger hoist; *2*—steel cable

Organ timbering consists of rows of posts (two or three) set up at some distance from each other or closely spaced. To secure quick and safe passage for workers special openings are left in organ timbering at every 5 metres (Fig. 59). Cog (or cluster pigsty) timbering replacing crib support consists of sticks. Each pigsty is made up of several sticks—four or six and more, put up near one another.

Before caving of the roof in the mined-out area both crib and organ timbering sustain increased rock pressure. The boundary of the caving area usually coincides with the line of crib timbering which facilitates the break of the roof rocks. Organ timbering is not shifted. Being able to withstand a considerable pressure of the superposed rocks, this support allows the cribs to be stripped and transferred to a new place.

Before starting artificial caving special timbering (transferred cribs and new rows of organ timbering) is set up as near as possible to the face breast with provision for the passage of a coal cutter

and the installation of a conveyer. In many cases cribs are not employed at all and use is made of organ timbering alone.

Artificial caving of roof with wooden timbering. In order to cave the roof the posts of the face timbering are knocked out with due observance of the prescribed safety regulations. When the supporting posts have been knocked out the roof collapses, being broken along the line of the organ timbering. Cribs are shifted to the face breast prior to pulling out the face timbering, while special organ timbering is neither knocked out nor shifted. Caving causes rupture in the solid mass of rock along the line of the organ timbering and the rock pressure at the face diminishes. As the stope advances these operations are repeated regularly.

Props of the face timbering should be knocked down by experienced workers in strict conformity with the established safety regulations. Only those props should be knocked down over which the rock has no fissures. The knocking down of props should be started from the boundary of the section towards freshly installed organ timbering. Some of the posts are left in place as safeguards. Hearing these posts crackle the workers must immediately seek shelter behind the organ timbering. By shifting cribs and knocking down props much precious timber is saved.

When it is impossible to knock down the supporting timber, for instance because of the hazard of a sudden roof collapse, the presence of graded rock in the worked-out area, the great length of the face or the small thickness of the seam, the roof of the mined-out space is caved by blasting, for which the necessary number of blastholes are drilled in the back of the seam and then fired.

In coal mines of the USSR timber props are often pulled out by special tugger hoists (machine-induced caving of the roof). The cable winds around the hoist drum and pulls out the props (Fig. 59). In a coal wall up to 100 m long artificial roof caving by this method takes no more than 20-25 minutes.

Among the major advantages of powered artificial caving of the roof are safety of the operations, and also complete and rapid caving.

The main shortcoming of powered caving is that the mine timber remains in the gob area, though part of it (up to 30%) can be recovered by hand before caving is started. Powered caving is ruled out when the coal wall is supported by metal posts.

In steeply pitching beds face and crib timbering is employed, and in addition to this the floor and roof are lagged with wooden slabs. In working steep seams knocking down of face timber is dangerous and is prohibited.

The use of steel posts and of powered support structures. To support productive workings in coal mines of the USSR metal posts of various designs are widely used.

Investigations of metal support posts carried on for many years and wide-scale tests conducted by a number of research institutes made it possible to find the most practical designs with special reference to various conditions of mining horizontal and slightly inclined seams. In recent years metal posts have been used in working low steep seams on an experimental scale.

The utilization of metal support in a number of coal basins of the USSR helped to appreciably cut down timber consumption and expenses on its transportation, improve the conditions of operation

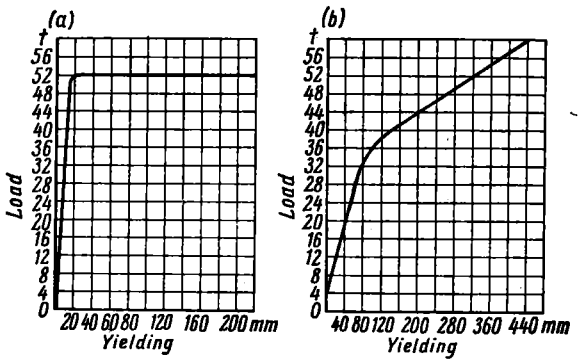


Fig. 60. Performance characteristics of metal posts:
a—posts of constant resistance; *b*—posts of progressively increasing resistance

of machines used in production faces, raise the efficiency of timbermen and reduce the prime cost of the coal mined.

The metal posts in use may be classified as rigid and yielding. In coal walls of gently sloping seams use is now made of yielding metal posts of progressively increasing and constant resistance.

Constant-resistance posts are becoming most popular, since they possess a number of advantages over posts of progressively increasing resistance: high initial resistance, maximal yielding without reduction in the resistance of the post, mechanical protection against overloading. The yielding of a post, i.e. its lengthwise contraction is needed to ensure uniform loading and protect the post against damage by the subsiding roof.

Figure 60 illustrates characteristics of metal posts of constant and progressively increasing resistance. Rigid posts are not considered, because they are rarely used.

Since 1957 improved individual hydraulic face posts have been employed in the mines of the Donets coal basin.

The latest designs of face posts are discussed below.

Yielding Metal Posts of Increasing Resistance

In these posts resistance increases gradually and reaches the level of the service load only after the extension part comes all the way down.

Below we present a brief description of one of the Soviet types of metal post (Fig. 61). The post consists of a base or body *1*, extension (telescopic) *2* and wedge lock *3*. The body, made of a special square section tube, has holes *4* disposed in two staggered rows. When installing the post, setting wedges are inserted into these holes underneath the extension, thus ensuring an adequate thrust of the post against the roof and floor. The extension is made in the shape of a wedge with the working edge sloping at 0°30′. The wedge lock consists of the upper and lower clamps, two side plates and hree wedges—two vertical and one horizontal. As the horizontal wedge is driven in the vertical wedges are thrust against the extension of the post and firmly

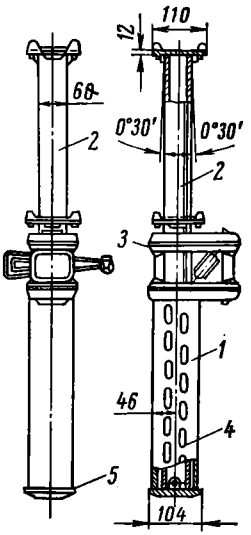


Fig. 61. General view of metal post for seams 0.8 to 1.8 m thick

grip it. To loosen the post the horizontal wedge is knocked out. In order to enlarge the surface of the post contacting the floor and the roof shoes *5* are welded at its ends.

Yielding Posts of Constant Resistance

In these posts the extension part begins to come down only when the rock pressure reaches the service load level. Until this happens the post is practically rigid. Further descent of the post extension takes place under constant resistance equalling the service load.

Yielding posts of constant resistance are exemplified by tubular and hydraulic posts. The extension part in posts of constant resistance is held in place by cohesive force or pressure of a fluid. The constancy of the friction forces in posts is achieved by eliminating the taper and by thorough processing of large friction surfaces.

Soviet tubular posts (Fig. 62) have a working resistance from 10 to 25 tons. The post is fixed in place by means of a thrust bolt.

TC-10 posts, for example, are issued in five types and sizes, their minimal height being from 325 to 700 mm and the weight from 8 to 13 kg. The minimum height of TC-25 posts ranges from 779 to 1,113 mm and the weight from 30 to 38.3 kg.

Hydraulic posts (Fig. 63) have a working resistance from 15 to 25 tons. They are employed in working gently dipping and inclined beds 0.65-2.0 m thick.

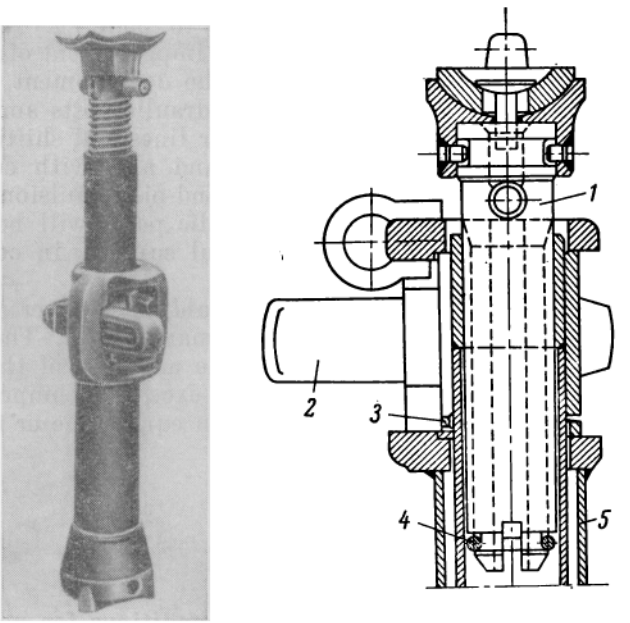


Fig. 62. A tubular post:
1—outward thrust screw; *2*—wedge; *3*—spring; *4*—ring; *5*—post body

The main parts of one of the types of hydraulic post are: cylinder, piston with rod, hand-operated pump, by-pass and relief valves. As the oil from the upper portion of the post is pumped to the lower portion the post begins to extend. When the roof subsides the load on the post increases and so does the pressure inside the working space. The oil pressure reaching a pre-set value, the relief valve opens, the oil from the working space flows along the pipe to the top portion of the post, which begins to descend, offering a constant resistance to the subsiding roof. Among the merits of these posts are their small weight, convenience of installation and recovery, since they can be pulled out remotely.

In recent years the Skochinsky Research Mining Institute and the Donets machine-building plant have conducted a number of investigations aimed at the improvement of hydraulic posts, such as designing of posts with a closed hydraulic system and a water-and-oil emulsion as a working fluid, the betterment of the design of the posts and cutting down their weight. (Posts of the ГСУ type have a working resistance of 20 tons, a minimal height within the range from 480 to 1,120 mm, extensibility from 170 to 630 mm and weight from 19.6 to 36.8 kg.) The prime objective of further improvement of hydraulic mine support is the development of sets of equipment with hydraulic posts and powered flitting of conveyer lines and shifting of the breaker support, and also with centralized supply of water-and-oil emulsion. In the near future hydraulic posts will become the main type of metal support in coal mines of the USSR.

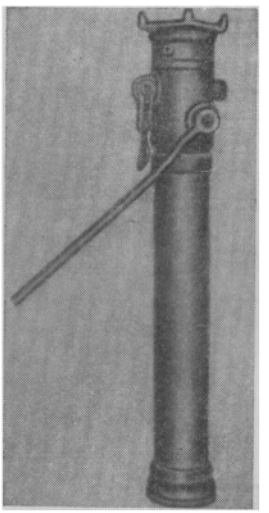


Fig. 63. A hydraulic post

Metal posts should be recovered completely and re-used many times. They should be shifted with the advance of the face so as to prevent their excessive compression (an artificial caving run equals one or two cuts).

To ensure more favourable operating conditions for metal posts it is practical to use them together with metal headboards. These are made of steel and are usually of split-swivel design 600-1,400 mm long. They are set up perpendicularly to the stope line and enable supporting the roof in the stope area and, hence, flitting the conveyer without dismantling it. The metal headboard weighs 15-60 kg, depending upon its size. Permissible load in the centre of the headboard is 30-35 tons, with 2-5 tons at the end of the cantilever. To secure the headboard in place one end of it is provided with a stay fork and the other with a lug.

The application of metal face posts reduces timber consumption on the average by 7 cu m per 1,000 tons of coal produced and cuts down running expenses by as much as 0.165 rouble per 1 ton of coal. Along with individual metal posts, metal breaker props, replacing wooden organ timbering, are being introduced on a wide scale in coal mines of the USSR.

Yielding props for induced caving of the ОКУ type (improved breaker timbering) find extensive application at present. The ОКУ-type prop (Fig. 64) consists of a stand, main screw, set screw and top plate. When setting up the prop under the roof the main screw is first moved up by turning it with small crow bars inserted into special holes, following which the key is driven in, which presses the thrust-strip against the set screw. The prop is fixed in place

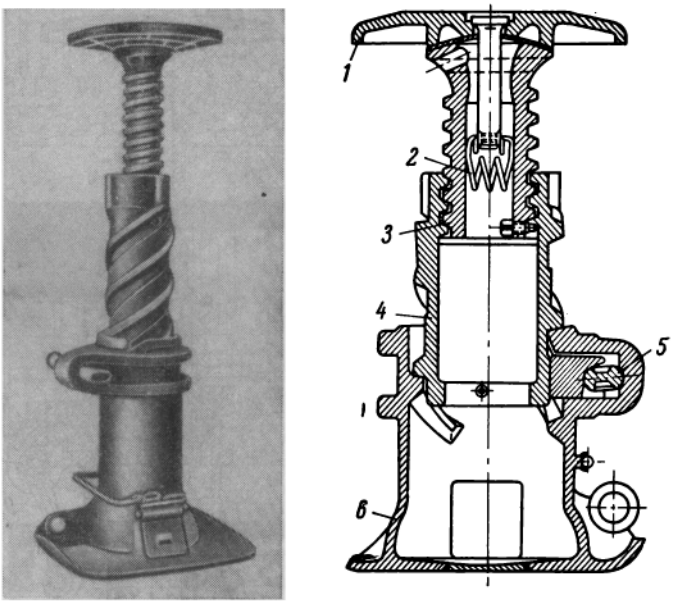


Fig. 64. ОКУ-type artificial-caving post:
1—top plate; *2*—spring; *3*—set screw; *4*—main screw; *5*—key; *6*—stand

by means of the set screw. In order to remove the prop the key is driven out. ОКУ-type props are manufactured in seven types and sizes for seams from 0.45 to 2.1 m thick.

Depending upon the type and size the maximum height of the props is 585-2,000 mm, the minimum 323-1,035 mm. The yielding capacity for the working resistance equals 40-140 mm. The working resistance is 100-200 tons. The prop weighs from 90 to 355 kg, depending upon the type and size.

The ОКУ props made it possible to introduce roof control by complete caving of coal walls, cut down timber consumption on the average by 22% and reduce labour required for roof control by 15%.

New and improved types of breaker post are undergoing tests in the field. Their advantage is a mechanism for remote pressure relief. By way of example, the ОКД-type post is shown in Fig. 65.

In recent years hydraulic breaker posts have come into use. So far as their performance characteristics are concerned they should be included in the category of constant resistance support. Their bearing capacity is 80-100 tons and more.

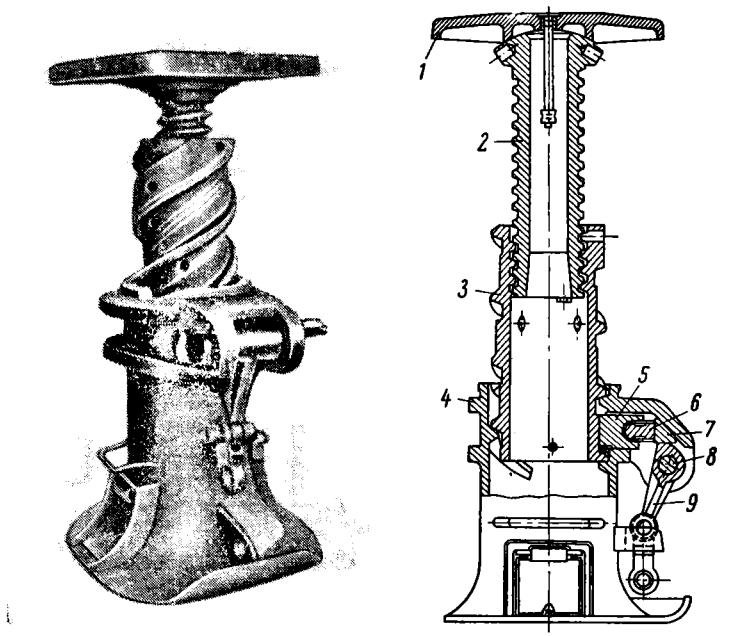


Fig. 65. ОКД-type artificial-caving post:
1—top plate; *2*—set screw; *3*—main screw; *4*—stand; *5*—block; *6*—horizontal key; *7*—intermediate key; *8*—shaft; *9*—cam shaft

A diagrammatic representation of a coal production face with metal support is given on one of the standard specifications for roof control compiled for the Donets basin mines (Fig. 66). This standard specification is a document containing technical solutions for all problems of roof control and production face support. Safety regulations stipulate that each production face must have an approved standard specification for roof control.

Along with individual metal support, sets of powered units and whole assemblies are being introduced in mines of the USSR. The use of these types of support ensures the maximum safety of opera-

Main characteristics	Units of measurement	Magnitude
Coal height (a) aggregate (b) minable	m m	1.0 1.0
Pitch angle of seam	deg	15
Length of face	m	150
Method of roof control	Complete caving	
Depth of cut	m	1.45
Width of pack strips (a) under ventilating entry (b) over haulage entry	m m	8.0 10.0
Caving interval	m	1.45
Face support density	Posts per 1 sq m	1.14

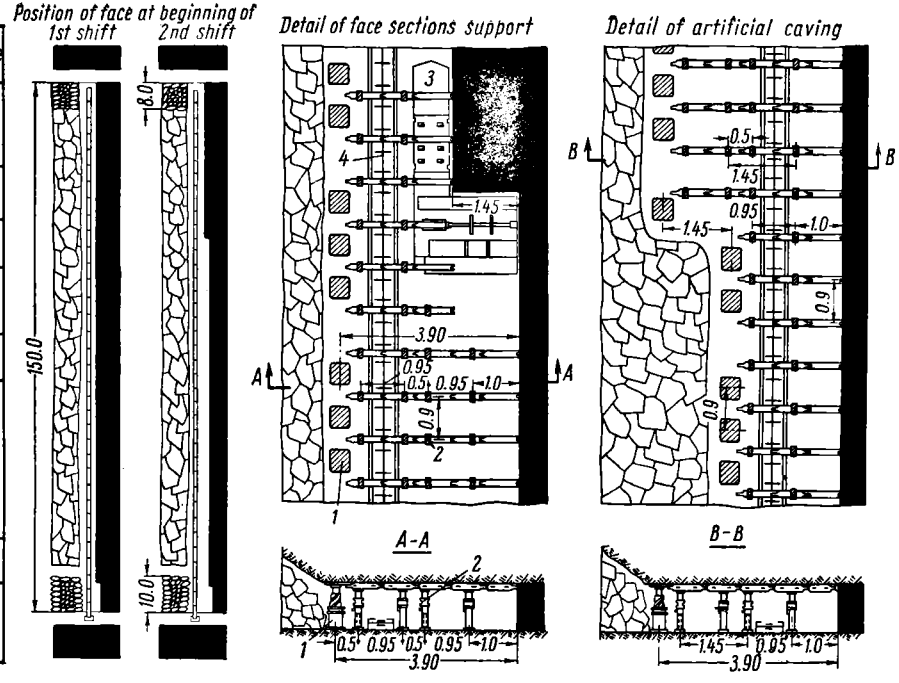


Fig. 66. Standard specification for roof control with support by means of OKY artificial-caving posts and metal posts with hinged headboards, as recommended by the Donets Coal Research Institute:

1—OKY-type artificial-caving post; 2—metal post with hinged headboard; 3—universal mining machine; 4—flight conveyer

tions, rapid advance of the production face and high labour efficiency.

The metal supporting set is a structure made up of metal posts connected by a common headpiece. A characteristic feature of a complex supporting set is the absence of braces between individual sections. Figure 67 illustrates a set of the КМ-9Д outfit, which incorporates a coal combine, an СП-46 mobile flight conveyer, an М-9Д-type supporting set, and a support shifter. This outfit services a coal wall up to 100 m long.

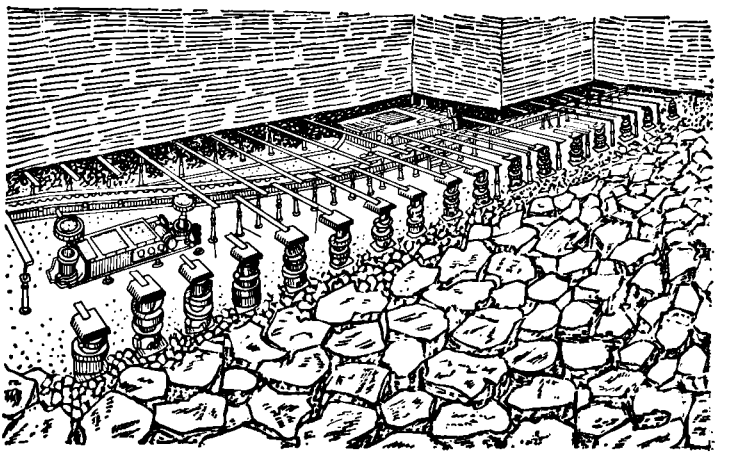


Fig. 67. A set of КМ-9Д outfit

Mobile supporting sets (shields) include units which closely follow the advancing stope; they are provided with bracings between sections (sets) of the support. The powered support usually makes part of a complex equipment set employed in stoping operations.

Powered sets used in the mines of the Moscow coal basin include, for instance, the mobile stoping unit ОМКТ "Tula" (Fig. 68). The МК set, operating in conjunction with a conveying worm narrow-range КШ-type continuous mining machine (Fig. 69), deserves special note. ОМТК and МК units are being successfully introduced in a number of mines of other coal basins of the USSR and in other countries. Experience shows that with the use of mobile supporting sets the coal output and labour efficiency in stopes are more than twice as high as at faces with ordinary supporting equipment. In some mines using powered supporting sets the face advance reaches 225 m per month, the output of the coal wall 69,000 tons per month,

the yield per faceman 1,250 t per month, and the yield per miner 132 tons per month. There is reason to expect a further improvement in the technical and economic characteristics of mines employing mobile supporting sets in stoping, particularly in respect of working conditions and safety of operation at coal walls.

Mobile metal (shield) support for steep seams has been designed by N. Chinakal, Corresponding Member of the Academy of Sciences of the USSR, and is successfully used in the Kuznetsk basin.

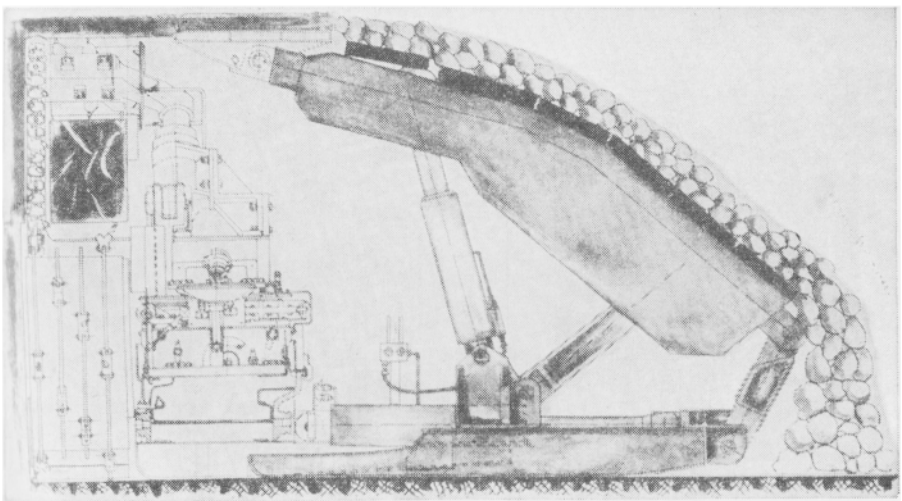
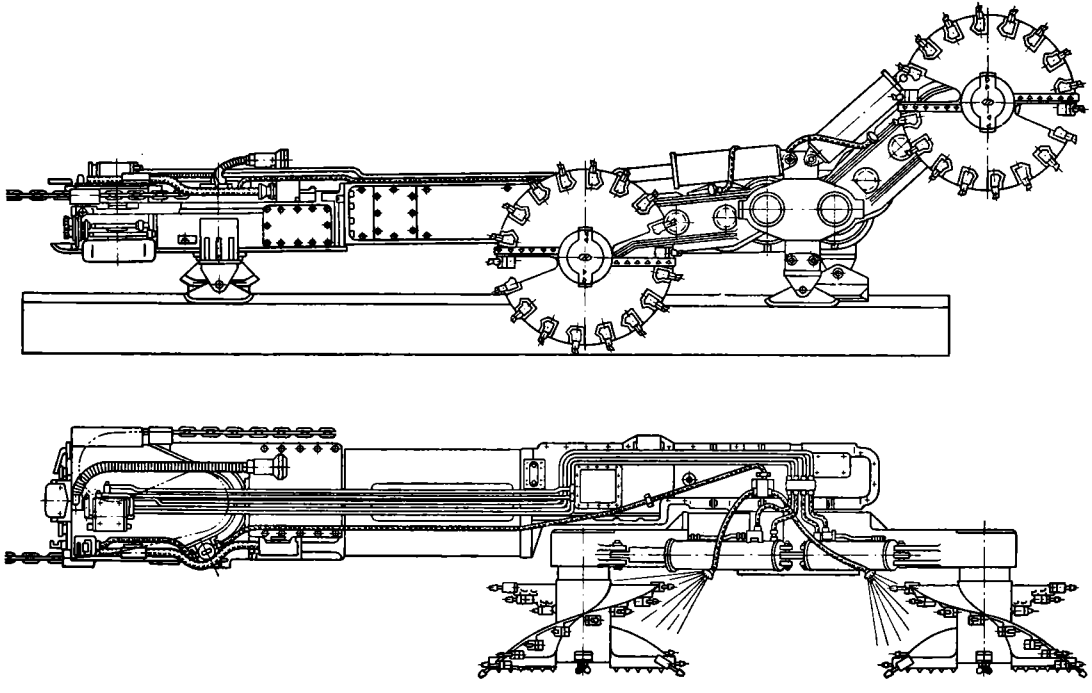


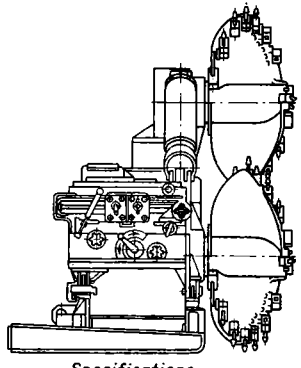
Fig. 68. OMKT-type mobile supporting set

The use of metal posts, breaker props and mobile powered supporting sets is one of the most important measures in mechanizing labour-consuming operations in rock pressure control.

The immediate task is the use of supporting sets making part of a combined extraction unit, which incorporates the support members, mining machine and conveyer. An example is the A-2 unit presented in Fig. 70. It is intended for coal walls 50-80 m long with slightly inclined beds 0.8-1.8 m thick. The unit consists of a planing machine or a narrow-range continuous mining machine, one-piece mobile drag conveyer and movable hydraulically operated support. Such units are controlled remotely (including shifting of the support) from workings adjoining the production face.

In the Moscow coal basin the construction of the Vostochno-Gryzlovskaya mine is underway. The mine is to be provided with mechanized sets for stoping operations, automatic control and





Specifications

1. Combine operates on shuttle principle from conveyer frame	
2. Output, tons/min	up to 3.3
3. Extracted seam height, m	1.5 – 2.5
4. Working range, m	0.5 – 0.8
5. Cutting speed (tooth penetration), m/sec	2.07
6. Type of feed – ГП41Ц, hydraulic feed assembly	
7. Feed rate, m/min	
(a) working	0 – 3
(b) light	0 – 10.5
8. Tractive effort, tons	
(a) working	10
(b) light	2.9
9. Vertical feed rate, m/min	0 – 5
10. Initial cutting feed rate, m/min	up to 0.2
11. Electric motor	
(a) Type	3 30 100/450 (ЭДК04-2с)
(b) Rated power, kW	100 (75)
(c) Revolutions per min	1460 (1470)
(d) Voltage, V	380/600 (380)
12. Working pressure in hydraulic system of operating mechanism, kg/cm^2	100
13. Basic dimensions of machine in operating position, mm	8500 × 940 × 1220
14. Weight of machine, kg	10642

Fig. 69. КШ-type continuous coal mining machine

regulation devices, entry-driving machines, integrated trains and gate conveyers. The output per miner is expected to reach 200 tons per month.

The next step will be the construction of a mine with all-round automation—a mining enterprise with a system of general electronic control from the ground surface, which will guide and regulate

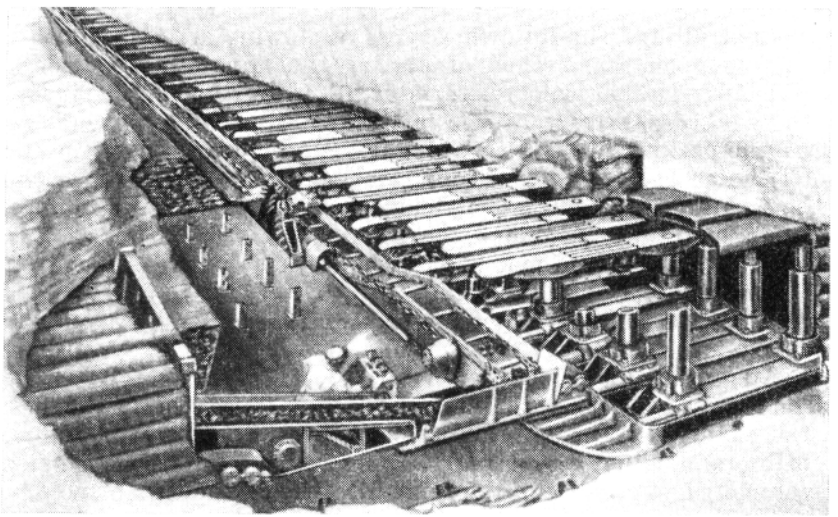


Fig. 70. A-2 unit

all mining operations. At that mine the workers will not directly participate in the extraction of coal. Their duties will be restricted to control and maintenance of the equipment, assembly, running repairs and replacement of worn-out parts.

4-4. FILLING OF MINED-OUT AREAS

Stowing or packing of the worked-out space is one of the methods of roof control. Stowing may be partial or complete, depending upon the extent to which the mined-out space is filled. The barren rock necessary for partial filling may be obtained directly in the mine from gangue intercalations when stoping, or when ripping or shooting-up the back and bottom of special, so-called dummy roadways, and also when driving sublevel and level entries, etc.

The barren rock for complete filling is obtained on the ground surface in special borrow pits. It is crushed down to a definite size and

passed along special stowage openings or boreholes down to the packing levels.

Barren rocks procured in waste-dumps, concentration plant tailings, smelter slags (especially granulated ones) may be utilized as mine-fill. The use of such materials greatly reduces the cost of filling.

Some information on stowing in coal and metalliferous mines is given below.

Types of filling. The following types of stowing are distinguished, depending upon the method of placing the mine-fill: by flushing; by slushing; mechanical, where mine-fill is thrown and compacted in the worked-out space by special throwing machines; pneumatic, where the packing material is brought to and compacted in the goaf under the effect of compressed air (with pneumatic filling the material may also be made to move along the workings by using compressed air); hydraulic, in which the packing material is delivered to the site for stowing and made to fill the worked-out area by a strong jet of water.

Filling with layers of dry material subsequently sprayed with water or pulp (a mixture of water and clay or sand) is termed wet to distinguish it from hydraulic filling.

The required amount of mine-fill in situ is estimated on the basis of the volume of the mined-out space (less workings not subject to filling and minus timbering), taking into acount the coefficient of volumetric expansion of the mine-fill. The coefficients of volumetric expansion for some materials are listed below:

Running fine sand	1.05
Sand, gravel	1.10-1.20
Sandy loam, loam, soft clay	1.20-1.25
Marl, humus	1.25-1.30
Hard, heavy clay	1.25-1.35
Soft rocks	1.30-1.40
Hard rocks	1.40-1.50

On filling the mined-out space the packing material becomes compacted. The shrinkage of the material depends on the properties of the rock, the thickness of the seam, the angle of dip and the stowing method. Shrinkage (P) is usually expressed in per cent and can be determined from the formula

$$P = \frac{V_1 - V_2}{V_1} 100\%$$

where V_1 = volume of the initially stowed space, cu m;
V_2 = volume of the stowed space following shrinkage of the mine-fill, cu m.

Shrinkage for different types of packing is:

Hydraulic (float) fill	5-12%
Pneumatic and mechanical filling	10-15%
Wet fill	15-20%
Dry fill by flushing in steeply dipping seams:	
with fine-grained rock	15-25%
with rock in big lumps	25-40%
Dry hand-stowing in gently sloping seams	40-50%

Use of Different Types of Filling

Filling by flushing is employed in mining steep coal seams and also when extracting ore in rooms or by inclined slices.

Stowage by slushing is used with mine-fill consisting of lumpy material of different size, including big lumps. The application of mechanical packing by scrapers in bedded coal deposits is very limited;

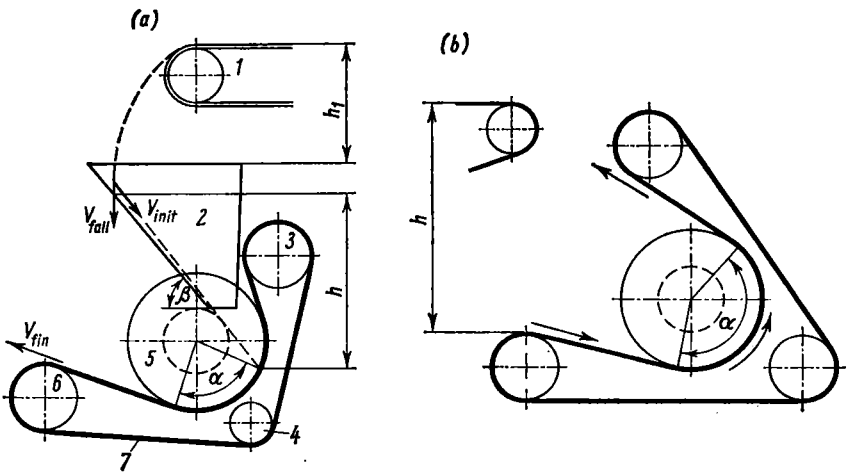


Fig. 71. Diagrams illustrating the operation of mine-fill throwing machines: *a*—with bottom ejection; *b*—with top ejection; *1*—feed conveyer; *2*—charging hopper; *3*—driving drum; *4*—idle guide drum; *5*—pressure drum; *6*—throw-off drum; *7*—belt

at present it is employed chiefly in working ore deposits by systems involving extraction of ore in horizontal slices.

Mechanical mine-fill-throwing machines are at present used in the Kuznetsk basin and have successfully stood the test in the chalcopyrite mines in the Urals. These machines throw off the mine-fill from an endless belt moving at a high speed. The material is fed onto the belt through a special hopper and a flanged drum.

We distinguish machines with top and bottom charging and ejection. The large angle at which the belt leaves the pressure drum in machines with top ejection ensures a high velocity of mine-fill throw, which explains their wide use. Diagrams illustrating the operation of the machine are shown in Fig. 71.

Specifications for the mine-fill throwing machines in use are listed in Table 12.

Table 12

Specifications for Mechanical Stowing Machines Employed at Present

Characteristics	Type of throwing machine	
	M3-1 with bottom ejection	M3-3 with top ejection
Rated capacity, m^3/hr	60	35
Belt speed, m/sec	15	20
Throwing distance, m	6-8	11.2
Rated power of electric motor, kW	11.4	16
Weight of machine with motor, kg	870	750
Width of belt, mm	500	500
Overall dimensions, mm:		
length	1,750	1,560
width	860	805
height	950	1,100
Slope of belt, deg	5-30	14-45

Throwing machines are intended mainly for operation in short working faces up to 10 m long, to move pack rocks 40-70 mm in size. When used at long faces they may be shifted on the frame of a face belt conveyer along the coal wall. This conveyer feeds the fill to the machine. The packing material is delivered to the thrower on the upper branch of the belt conveyer. The machine is placed underneath the upper branch from which the fill is discharged into the receiving hopper of the thrower by means of a plough-type raker.

In short faces the solid mass of pack is built up in ribs 6-8 m wide. On the side of the working face the goaf is fenced off with lagging. Pilot tests conducted in coal mines prove the possibility of using hydraulic transport of mine-fill to the throwing machine. In this case the delivered material should previously be dewatered.

Among the advantages of the mine-fill-throwing machines are their ability to bring the pack up to the roof of the stoped-out area and relatively low power consumption (0.4-0.5 kW-h per 1 cu m of

the fill). Their drawbacks include: rigid limits on the size of the material moved, a small throwing distance, low compaction of the pack (shrinkage 20-30%), difficulty of shifting the machine and high wear and tear of the belt (it has to be replaced after a passage of 1.5-3.5 thous. cu m of the fill). Because of these shortcomings the use of the mine-fill-throwing machines is rather limited.

With pneumatic filling the required throwing velocity is achieved by using the general principle of pneumatic transportation, viz: a body entrained by a current of air is subjected to its pressure, which is the greater the higher the velocity of the current.

Two instances of pneumatic filling should be distinguished. In the first, the current of compressed air is utilized exclusively for throwing lumps of mine-fill. A stowing machine with a short nozzle is set up near the space to be packed in the same way as mechanical throwing machines. The mine-fill is delivered to the machine by ordinary conveyers. Such units, called ejector-type stowing machines, have not found practical use, since their operating characteristics are much inferior to those of the mechanical throwing units described previously.

In the second case, the air is used for pneumatic transportation of the packing material along pipes and for throwing it into the worked-out space. Stowing machines (of a semi-stationary type) employed in this case are usually set up on the boundary line of the producing section. The mine-fill is delivered by ordinary conveyers, fed into a stowing pipeline up to several hundred meters long and conveyed along it to the work-place. The pipe diameter is 150-250 mm.

The maximum size of the mine-fill should not exceed half of the pipeline diameter. Two types of such semi-stationary stowing machines are in use: chamber-type machines with a proportioning cell-type revolving wheel ПЗМ-1; chamberless machines: МПЗМ-1 with a level drum and БПЗМ-2 with an upright drum (Fig. 72).

Stowing machines with a proportioning drum are simpler in design and smaller in size. Specifications of pneumatic stowing machines are given below:

	МПЗМ-1	БПЗМ-2	ПЗМ-1
Rated capacity, m^3/hr	up to 40	80	up to 35
Mine-fill transportation distance, m	up to 400	400	600-800
Air consumption per 1 cu m of fill, cu m	80-160	100-130	120-180
Diameter of stowing pipeline, mm	150	150	150
Air pressure in stowing pipeline, atm	up to 2.5	2.5	up to 4
Overall dimensions of the machine, mm			
length	2,650	3,300	2,500
width	670	1,600	1,760
height	1,010	1,500	2,300

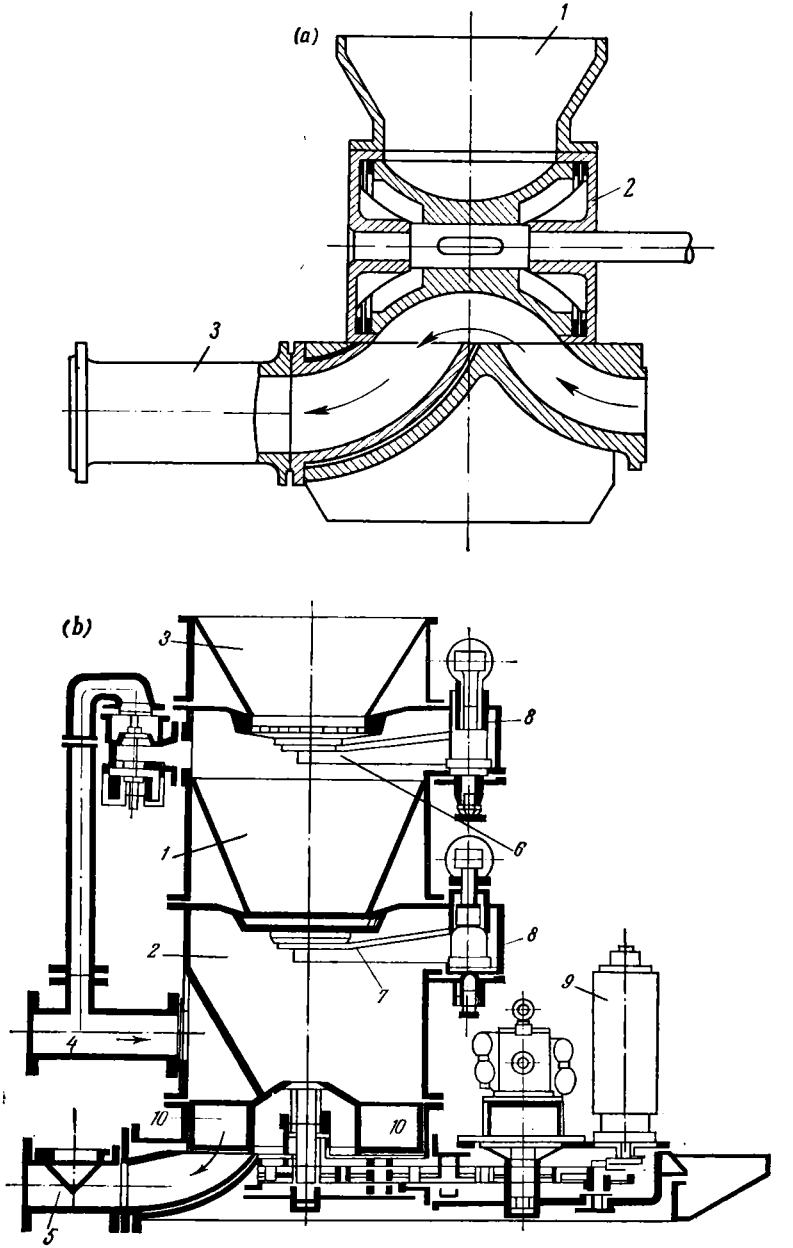


Fig. 72. Pneumatic stowing units:

a—МПЗМ-1 pneumatic goaf-stowing machine: *1*—receiving hopper; *2*—proportioning cylinder; *3*—mine-fill delivery pipe (arrow indicates direction of compressed air flow); *b*—ПЗМ-1 double-chamber stowing machine: *1,2*—chambers; *3*—receiving hopper; *4*—air pipeline; *5*—flushing pipe; *6,7*—opening and closing gates; *8*—air-cylinders (for operating the gates); *9*—distributor; *10*—cell-type revolving proportioning wheel (arrow indicates direction of compressed air flow)

Among the merits of pneumatic filling are: the high compaction of the fill, simplicity of handling, supplementary aeration of mine workings by return compressed air.

The disadvantages of pneumatic filling include: considerable initial outlays for the equipment (compressor units, pipelines), high power consumption, rapid wear of pipes, limited number of pipeline elbow bends (up to four-five) and excessive dust formation during stowing. The dust can be appreciably reduced by introducing water into the pipeline at a distance of 20-30 m from the discharge end.

Experience shows that for packing material with a minimum particle size of 50-80 mm the inner diameter of the stowing pipeline should be 150-200 mm. For trunk lines steel pipes with walls up to 10 mm thick are employed; pipelines within the working face area should have walls 4-6 mm thick. Pipes with four elbows wear out completely after the passage of 30 thous. cu m of clay shale or 15 thous. cu m of sandstone. When lined with inserts of cast basalt the pipes are capable of passing up to 1 mil. cu m of fill.

Among the new methods of stowing at mines mention should be made of filling with the use of the power developed by ВВД-8 or ВВД-9 air jet of booster fans, or of compressed air directly from the general air-network. Large-scale experiments carried out at one of the Soviet mines in 1955-1957 have proved this method of stowing to be more effective than hand-packing. In installations of this type stowing machines are simple and versatile, the capacity of a unit with a fill-delivery distance of up to 40-60 m (particle size from −80 to +10 mm) is up to 4-5 cu m per 1 hr of operation. Air consumption with the use of mine fans reaches 120 cu m per 1 cu m of fill, and with compressed air taken directly from the general air-network up to 160 cu m per 1 cu m of fill. The net cost of stowing is 40-50% lower than with hand-packing.

Of prime importance in raising the effectiveness of stowing with the aid of a compressed air jet is batchwise ("cannon-shot") delivery of air and fill. This should enable air consumption to be cut down to 60-70 cu m per 1 cu m of fill.

One of the most effective methods of stowing is hydraulic filling, which has found wide use in some coal deposits of the USSR, Poland, the Upper Silesian basin, Hungary, Rumania, France and in many metalliferous mines of Canada, USA, France and other countries.

The principles of hydraulic stowing consist in the following (Fig. 73). With the aid of hydraulic monitors the filling material (tailings, slag, crushed rock) is first mixed with water, following which the pulp (mixture of the material with water) flows by gravity along a vertical or inclined pipeline down to one of the levels and then under natural head proceeds through a level pipeline to the rooms or blocks to be stowed.

From the level pipeline the fill is delivered through branch pipes and a hose to the open goaf where the solid particles settle down and fill the goaf by gravity. On sedimentation of solid particles the water

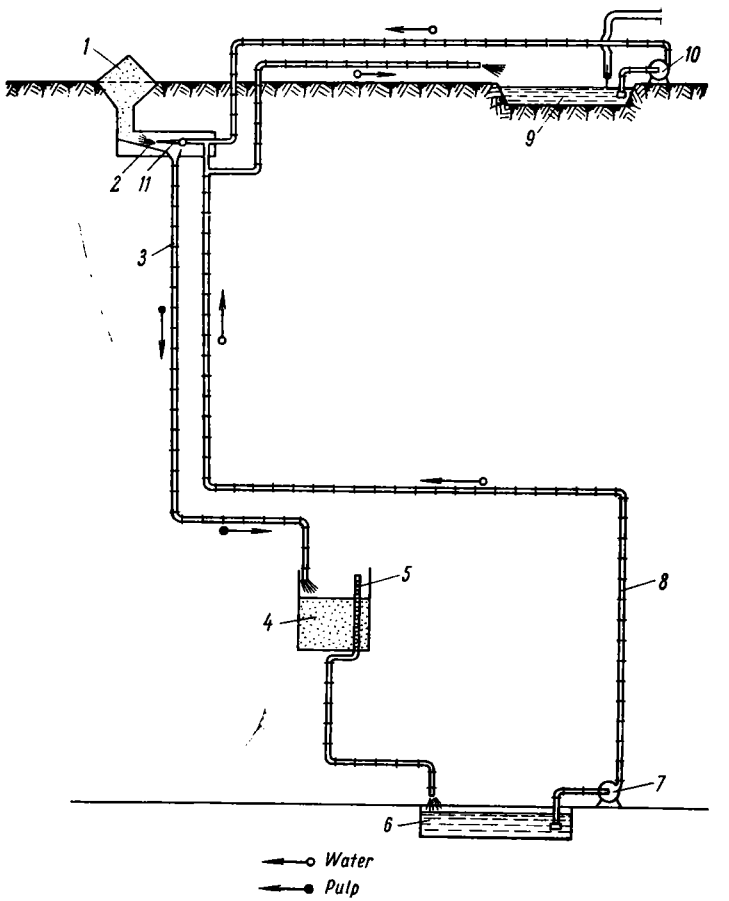


Fig. 73. Skeleton flow-sheet of hydraulic filling:
1—storage bin for mine-fill; *2*—mixing trough; *3*—flushing pulp pipeline; *4*—solid fill mass; *5*—drain pipe; *6*—underground water settling tank; *7*—pump; *8*—pipeline; *9*—surface water settling tank; *10*—pump; *11*—mixing chamber with slushing giant

is passed through frame filter partitions to the corresponding workings and runs along the outfall ditches into a water collector, whence it is pumped to the ground surface and can be re-used after clearing.

Hydraulic filling is widely used at chalcopyrite mines in the Urals.

The mixing chamber is set up at an elevation mark which should ensure the pressure head needed for the transport of the material along the level underground pipeline without clogging it. At copper mines in the Urals the ratio between the length of the upright and that of the level pipeline fluctuates from 1 : 0.7 to 1 : 3, depending upon the mining depth.

In the mixing chamber the mine-fill (granulated slag) is fed uniformly through a gauge box and a special chute onto a trough at an angle of 15-18°. From the trough a water jet discharged by a hydraulic giant washes the material off at a pressure of 4.5-5 atm into a steel funnel 800-1,000 mm in diameter. The funnel is covered with a steel grating 6-8 mm thick, having 35-50 mm holes. The purpose of the funnel is to stop oversize lumps and chips. In coal mines the stowing material is often subjected to preliminary screening to separate big lumps from fines less than 5 mm in size. In order to secure uniform delivery of the mixture into the pulpline additional water is delivered to the grating under a constant pressure from a special tank. The pulpline is made of cast iron or steel piping 90-125 mm in diameter and 8-12 mm thick.

Before they wear out cast iron pipes pass 3.5-4 thous. cu m and steel pipes up to 15-20 thous. cu m of granulated slag. With sand and clay material the pipes may serve much longer, the amount of material passed through them reaching 50-100 thous. cu m. The volumetric ratio of solids to water in the pulp is 1 : 2-1 : 3.

In the Urals the capacity of one hydraulic filling installation varies between 15 and 25 cu m per hour. The efficiency of stowing units at other enterprises reaches 40-60 cu m per hour with crushed rock, 90 cu m with slags and concentration mill tailings and 200-300 cu m with sand flushing. To seal off the space to be stowed, chutes and round timber flooring 10-12 cm thick are made in the block after extraction of one slice, the flooring being covered with 25 mm thick boards over the entire length and breadth of the block, then all slots and chinks are tightly stuffed with hay. Prior to stowing the corresponding slice, the chutes and the upraise walls adjoining them are lagged off from outside with overlapping boards and the chinks tightly filled with hay.

Among the merits of hydraulic filling are: the high compaction of the fill, high stowing efficiency, simplicity of operation, low initial outlays for installation, low net cost of stowing. As compared to hand-packing, the prime cost of hydraulic filling is reduced by as much as 1.5-2 times. The major item in the prime cost is expenses for procuring mine-fill, which make up 60-65% of the entire cost of hydraulic stowing.

The disadvantages of hydraulic stowing include: high water inflow into mine workings, entrainment of small mine-fill fractions

into haulageways together with water, the necessity of erecting water-settling tanks and bulkheads for fencing off the space to be stowed.

The basic steps to improve hydraulic filling are: the betterment of the pulp-formation process, including the building up of a closed system of water movement in the mine; more rational methods of water settling and clearing through the use of improved settling tanks, coagulation and special filters; automation of key operations in hydraulic stowing.

The ultimate criterion of the efficiency of stowing is the net cost of 1 cu m of the fill in place. The practical experience gained at Kuznetsk coal fields, where different filling methods are employed, yields the following data:

Table 13

Methods of stowing	Labour required for stowing per 100 tons of mine output, man/shift	Stowing unit capacity, m^3/hr	Comparative cost per 1 cu m of fill in place
By gravity	56	up to 30-40	0.8-0.9
Mechanical (throwing machines)	136	13-20	1.13
Pneumatic	86	30-35	1.33
Hydraulic	77	55-60	1.0

In comparing different types of stowing one has to take into account the compaction of the mine-fill (maximal with hydraulic fill and minimal with flushing).

In another method using self-cementing fill the open goaf is stowed with a hydraulic mixture, which hardens, becomes monolithic and tough with the passage of time (its compressive strength reaches 50-60 kg/cm^2 and more). In the USSR self-cementing fill was introduced only recently; however, owing to a number of important advantages its use in stowing increases year by year. Self-cementing fill is gaining wide application in a number of mines of the USSR and some foreign countries (the GDR, Poland, Finland, Canada, the USA, and others). Among its merits are:

(1) the possibility of simultaneous mining of deposits by underground and opencast methods;

(2) favourable conditions of working self-igniting ores;

(3) the absence of sagging of the ground surface underneath water bodies, buildings and plants;

(4) the possibility of effective robbing of previously abandoned pillars of valuable ore;

(5) the possibility of advanced mining of the most valuable ores and prolonged conservation of less valuable ones;

(6) the possibility of drastic curtailment of losses in extracting valuable ores by cut-and-fill systems, or by rooms with subsequent stowing.

A mixture of binding materials and inert aggregates (usually sand) is employed as mine-fill. Sometimes small amounts of plasticizers (clay, limestone) and hardening accelerators (lime, gypsum, etc.) are added to the mixture.

Low-grade cements (slag cements grade 125-225 kg/cm^2 or Portland cement of a grade not higher than 350 kg/cm^2), ground blast-furnace slags or power station electric filter cinders are used as binding agents. In the USSR ground smelter slags are frequently employed. The slag quality factor is determined by the formula:

$$F_q = \frac{\%\ CaO + \%\ Al_2O_3 + \%\ MgO}{\%\ SiO_2 + \%\ MgO}$$

Blast-furnace slags with $F_q > 1.6$ are fit for use as binders. In addition to F_q it is necessary to determine the modulus of basicity and the modulus of slag activity.

The basicity modulus equals:

$$M_b = \frac{\%\ CaO + \%\ MgO}{\%\ SiO_2 + \%\ Al_2O_3}$$

The basicity modulus should exceed unity (then no activating additives needed).

The activity modulus M_a is:

$$M_a = \frac{\%\ Al_2O_3}{\%\ SiO_2}$$

The value of M_a should be not less than 0.20, otherwise specially fine grinding and addition of hardening accelerators will be required.

Power station cinder trapped by electric filters is a good binder. A mixture of cinders and cement (unground) may also be used.

To bring the strength of the packing block in situ to 50-100 kg/cm^2 (after 3-6 months) the proportion of cement in the binder must be 20-30%.

The composition of the mixture for self-cementing fill may be formulated approximately by adopting the following weight ratio of the components:

$$C : A : I = 1 : (0.5\text{-}1.5) : 6;$$

where C = cement (binder);
A = additive (usually clay);
I = inert aggregate (sand).

This ratio should be adjusted after testing mine-fill samples for strength.

The hydraulic mixture of the self-cementing fill should also possess the necessary plasticity to ensure its free movement. The plasticity of stowing material transported via pipelines must be as great as 9-12 cm (the depth of submersion in the grout of a standard metal cone weighing 300 g, 15 cm high, with an angle of 30° at the apex). Excessive plasticity (high water-binder ratio) affects the strength of the fill and is therefore inadmissible. Strength (compression) tests are made on cubic specimens measuring 7 × 7 × 7 cm cast in metal moulds, taking 15 specimens from each batch. At least 3 specimens are tested at definite dates (after 28 days; 2,3 and 6 months).

A typical flow-sheet for the preparation of self-setting grout is shown in Fig. 74 referring to the Gaisk copper mine (Southern Urals, USSR), which successfully uses mine-fill composed of 360 kg of finely ground granulated slag, 27-40 kg of M-400 grade cement, 1,260 kg of sand and 320-340 litres of water per 1 cu m of fill.

The diameter of slushing pipelines is adopted within the range from 150 to 300 mm. The ratio between the maximum size of the inert aggregate particles and the pipeline diameter should lie between 1 : 4 and 1 : 5. The greatest distance of travel of the self-setting grout along the level pipeline may roughly be taken as a 3-4-fold height of the upright pipe set. L_{horiz} can be determined more precisely according to the formula:

$$L_{horiz} = \frac{H}{10}\left(\frac{\gamma}{\Delta p} - 10\right) - L_{\partial}$$

where H = upright section of the pipeline, m;
γ = specific weight of the grout (usually 2);
Δp = specific resistance of the pipeline to grout movement, atm/metres (ordinarily 0.05-0.07 for slag-and-sand grouts with 8-12 cm plasticity);
L_{∂} = equivalent resistance of elbow bends, may be taken as 15 m with an angle of 90°, 8 m with 45°, and 5 m with 30°.

When the flow of the slushing grout into the pipeline is stopped for more than 1 shift the pipes should be cleaned with water and compressed air.

After 3-6 months the solid mass of stow in place acquires a strength of 50-100 kg/cm^2, depending upon its composition, and can be exposed over 500-700 sq m without appreciable disintegration.

All the other types of filling described above are less costly, but their quality is much lower and in many instances cannot meet the

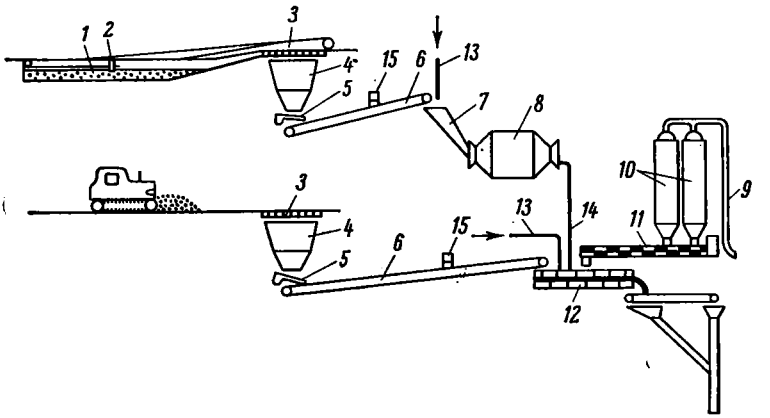


Fig. 74. Preparation of self-setting grout at the Gaisk mine:
1—trench-type storage place for granulated blast-furnace slags; *2*—slusher; *3*—grizzly; *4*—discharge bin; *5*—vibrating feeder; *6*—conveyer: *7*—feeder; *8*—ball mill; *9*—charging pipe; *10*—cement bins; *11*—worm feeder; *12*—C-543 continuous double-roll worm-type mixer; *13*—water supply; *14*—spout for delivery of pulp to mixer; *15*—ATM automatic scales;

specifications. Wide utilization of self-cementing stowing is one of the major technical trends in improving the mining of valuable minerals.

4-5. SYSTEMS FOR UNDERGROUND MINING OF COAL DEPOSITS

Coal deposits in the USSR are characterized by a great variety of coal beds in respect of thickness, angle of dip, water abundance, gas-bearing capacity, hardness and toughness of the coal, physical and mechanical properties and stability of wall rocks. This greatly influences the choice of the mining method, machinery and equipment. A large number of worked Soviet coal deposits include beds with gently sloping angles of dip (about 70% of all seams) and a limited thickness (1.5 m on the average).

A considerable proportion of coal mines present dust and gas hazards.

In the majority of coal fields the bottom and back of the seams consist of medium-rigid and medium-stable rocks.

In coal mines of the USSR mining systems involve extraction of coal by long faces (longwalls), in low seams (breast stoping or continuous mining), and in beds of medium thickness.

Long-pillar mining on strike has become most popular in post-war years.

In gently sloping and inclined thick beds slicing methods with the coal extracted in slices by longwalls are now most common, while in steeply dipping thick seams the shield mining method with caving of the roof rock predominates. Extraction of coal by long faces (longwalls) has promoted extensive mechanization of all coal mining operations.

The average length of the working face in coal mines of the USSR is at present 100-105 m, reaching in some coal fields 135-140 m (Pechora) and 165-170 m (Karaganda). The average monthly advance of a coal wall is 35-40 m. The advance of working faces at individual mines, however, appreciably surpasses the average figures, rising up to 50-60 metres per month and more. Considerable advance is recorded in operating with mechanized sets. In the near future the general trend in the development of the technology of coal mining by underground methods in the USSR will be working of seams by long working faces (longwalls) with the use of narrow-range combined coal-winning machines and coal planers, powered support sets, mobile flexible conveyers and other equipment integrated into mechanized complex sets. The extent of mechanization and the scope of mining operations in the mines of the USSR may be judged by the number of mechanical units, active working faces and their aggregate length as of the end of 1967 (for the entire coal industry of the USSR):

Movable powered support sets	550
Coal-winning combines	4,169
Coal planers	100
Entry-driving machines	604
Flight conveyers	42,316
Belt conveyers	13,384
Electric mine locomotives	14,763
Number of active working faces	4,837
Total length of active working faces	525 km
Average monthly advance of the active working face line	33.1 m
Average daily output per 1 working face	231 tons

4-6. BREAST-STOPING METHODS OF MINING

General

The term breast-stoping is applied to a method of mining in which the extraction of the mineral in a level is done by carrying a continuous heading without driving any development workings ahead of the production face or with a minimum amount of such work.

Conditions for Application of Breast-Stoping and Its Variants

The basic conditions warranting the application of breast-stoping include: predominantly stable wall rocks (in individual instances breast-stoping is practised with insufficiently stable rocks); a small and, more seldom, medium thickness of the bed; a gently sloping-to-steep angle of pitch.

Breast-stoping is often used in working coal deposits containing large amounts of methane, since with this system the number of uphill openings, where methane ordinarily accumulates, is very small.

With breast-stoping the mineral can be extracted both along the strike and up-raise. Mining along the strike is preferred, therefore the up-raise variant is not discussed here.

Extraction along the strike may be conducted with and without dividing the level into sublevels.

As regards the disposition of a gravity incline with low and moderate dip distinction is made between extraction with gravity inclines made in the worked-out area and working with advance gravity inclines driven in the solid mass of coal in place. In point of fact the latter variant is a compromise between the continuous and pillar mining systems.

Extraction starting from the gravity incline can be done with unilateral and bilateral gravity inclines.

Continuous Mining in Gently Dipping and Inclined Beds

Principles and their implementation. Let us consider continuous mining with a level-wise pattern of development, i.e. the variant involving the division of a level into sublevels with a unilateral gravity incline driven in the mined-out area (Fig. 75). In this variant a level with an interval 200-400 m high is mined by dividing it into gravity incline sections, each 400-500 m long. The sections are extracted by driving individual sublevels with the subjacent sublevel pushed forward 10-20 m ahead of the overlying one. No development openings are made in the solid mass of coal, except for an airway driven during the mining of the upper level and of a haulage entry running 50-100 m in advance of the working face. To deliver the coal broken in top sublevels a gravity incline with a manway and sub-entries are made in the worked-out space by shooting up the roof rock.

Gravity inclines with manways are run every 400-500 m so as to provide for timely replacement of the previously driven ones. Sub-entries are made concurrently with stoping and are gobbed up

after gravity incline sections have been worked out. Maintenance of openings made in the mined-out area is possible only by laying down packwalls near them (Fig. 75) or by abandoning protective coal pillars.

A variant of the system with a gravity incline in the solid mass of coal made in advance of the working face (Fig. 76) is applied when the drivage and maintenance of the incline in the worked-out space

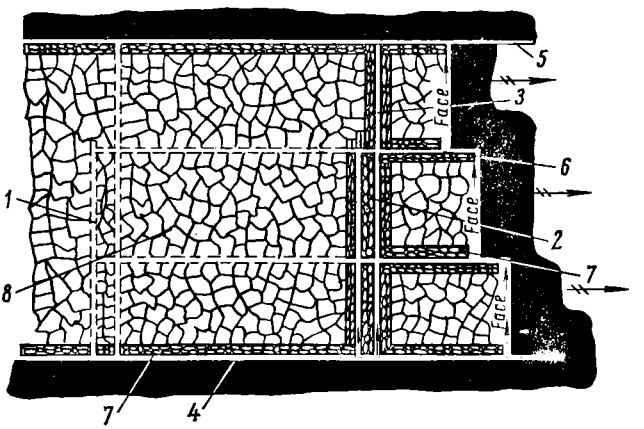


Fig. 75. Continuous mining with unilateral brake incline in worked-out space (timbering not shown):

1—gobbed gravity incline; *2*—active brake incline; *3*—manway for miner servicing the incline; *4*—haulageway; *5*—airway; *6*—subentry; *7*—packwall; *8*—worked-out space with caved roof; arrows indicate general direction of mining; ordinary arrows show air current flow

are difficult. In Fig. 76 the new gravity incline is shown in the stage of drivage. A manway and a holing chute are driven concurrently with it.

In a variant of the system with an advance gravity incline in the solid coal the haulage entry is driven in advance so as to ensure the timely drivage of a gravity incline intended for servicing the next producing section. With this variant, stoping in the extraction area is started from a special holing chute at a considerable distance from the gravity incline. An important shortcoming of this variant is the complicated ventilation arrangement, this being of particular significance in the presence of firedamp. In non-gassy mines the described variant is quite common.

Should it be necessary to step up the output of the extraction section, this may be worked on both sides of the gravity incline (Fig. 77). In this case the aggregate stope footage will be twice as great as in mining with unilateral gravity inclines. In bilateral gravity inclines the length of their producing sections will be appro-

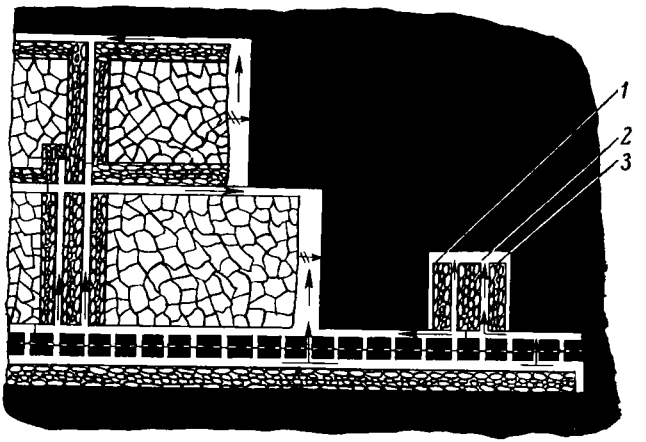


Fig. 76. Continuous longwall-level mining with an advance brake incline:
1—brake incline; *2*—manway; *3*—holing chute; arrow ⫲→ indicates direction of stoping, ordinary arrow shows direction of air current flow

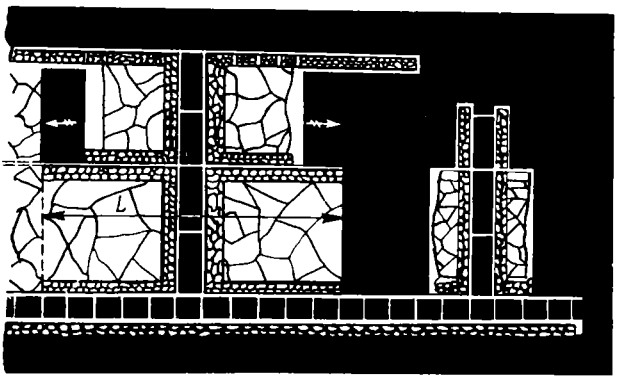


Fig. 77. Breast-stoping with bilateral gravity incline:
l—length of producing section serviced by the gravity incline; arrow indicates direction of stoping

ximately twice that of the unilateral ones. Bilateral gravity inclines are adopted with mine-fields of considerable extent along the strike.

The drawbacks of the breast-stoping variant with bilateral gravity inclines are: heavy traffic in the gravity incline, the reverse flow of the loads in one of the wings of the extraction area (relative to the general direction of the loads in the wing of the mine-field).

With regular occurrence of the bed and in the absence of geological dislocations and faults it is expedient to work the level by one continuous longwall in one level, without dividing it into gravity incline sections. This alternative (Fig. 78) has considerable advantages, such as: simpli-

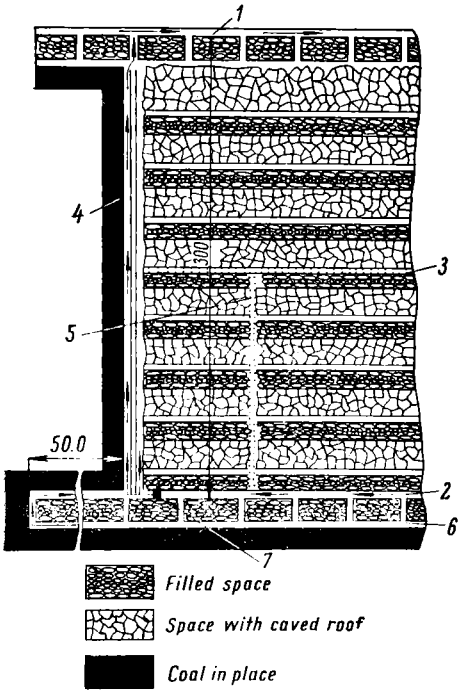


Fig. 78. The "longwall-level" version of continuous mining (in the plane of the bed):
1—ventilating entry; *2*—haulage entry; *3*—dummy roadway for obtaining mine-fill; *4*—active coal wall; *5*—emergency manway (with a high level interval); *6*—back entry used for aeration in entry driving; *7*—fan for secondary ventilation; arrows indicate direction of air current

city of operations, the small number of development workings, favourable conditions for maximal utilization of mining and transport machinery, no expenses for the maintenance of intermediate entries, better conditions for control over mining operations.

The choice of the working face length is limited by the permissible air velocity, difficulties in providing passage for men and supply of materials along an extensive face. In a long working face an appreciable volume of air is required, while its velocity cannot surpass the established standard (4 m/sec). According to the Safety Rules working faces must have two exits to the lower and upper entries and, when extremely long, a third one, in the middle of the stope. For this reason the length of a coal wall is ordinarily taken to be from 200 to 300 m. In other variants of the system the length of

the working face, as well as the interval height of sublevels is brought in line with requirement ensuring the optimal efficiency of mining and transport equipment.

Below is some additional information on the above-discussed variants of breast-stoping.

In breast-stoping intermediate entries are preserved by employing two methods (Fig. 79): abandonment of protective pillars of the mineral pierced for the purpose of communication and ventilation by small-sized upraises—holing chutes, or constructing a packwall built up of wall rocks shot up and ripped during the drivage of entries,

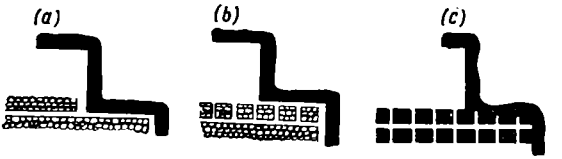


Fig. 79. Protection of intermediate entry (in the plane of the bed):
a—entry with barren rock packing strips; *b*—entry with pack pillars built of barren rock; *c*—entry with coal pillars

leaving the necessary passageways. Abandonment of mineral pillars near the entries entails substantial disadvantages, such as the necessity of excavating costly holing chutes and through-cuts, complicated mucking of the rock broken off in entries, considerable losses and dilution of coal during the subsequent robbing of pillars, difficulties in operating mining machines and conveyers. For this reason entry chain or stub pillars are rarely left. To safeguard haulageways against the considerable rock pressure they are driven with a narrow face and still pillars. The size of pillars is contingent upon the thickness and dip angle of the bed, the coal rigidity, the properties of the host rocks, and the mining depth.

Tentative data on the size of protective pillars near main haulage and ventilation entries in mining gently sloping and inclined seams are listed in Table 14.

A haulageway should be driven at least 50 m ahead of the working face so as to provide for spotting a train of mine-cars for coal loading. In mining with an advance gravity incline this distance should be accordingly increased.

In breast-stoping driftage of entries is a relatively simple matter. It is more difficult to drive gravity inclines with manways in the mined-out space. A gravity incline 3-4 m wide and 2.5 m high is cut by several headings driven from intermediate entries. Barren rock obtained by ripping the roof is laid down near the walls of the gravity incline and of its manway. In the alternative with the use of sublevels the distance covered by gravity inclines is the longer,

Table 14

The Size of Stub Pillars on the Dip in Working Gently Sloping and Inclined Seams, m

Mining depth, m	Wall rocks					
	Strong		Medium strong		Soft	
	Dip					
	Gently sloping	Inclined	Gently sloping	Inclined	Gently sloping	Inclined
Up to 200	15	12	20	15	25	18
300	20	15	25	18	30	20
400	25	18	30	20	35	23
500	30	20	35	25	40	25

the more expensive the drivage and equipment of the gravity incline, and it is the smaller, the higher the cost of maintaining intermediate entries and of the haulage along them. To reduce expenses and raise transport efficiency along intermediate entries use is made of mechanical or conveyer delivery, which is most promising, especially when conveyer lines are set up along the gravity incline. In all mine-openings connecting working faces with main haulageways it is practical to employ conveyer delivery.

In working easily dipping seams it is important to determine the rational length of the coal wall. In the practice of mining coal deposits this length has been gradually increasing and today has reached sizable proportions. For example, the overwhelming majority of mines in the Donets basin, where coal combines operate, have walls over 130 m long, and in the "longwall-level" variant of continuous mining 250 m and more. As their length increases the technical and economic performance at coal walls shows appreciable improvement, including increased labour efficiency, reduced volume of development workings per ton of the mineral output and a higher rate of advance of the headings. The factors limiting the length of coal walls are the capacity of the machines employed at the working face and the maximal permissible velocity of the air current. With long walls, roof control, including complete artificial caving, presents no difficulties.

The broad utilization of highly productive stoping machines and the possibilities of effective ventilation justify the present tendency towards a greater length of coal walls.

Major stoping operations and their mechanization. Stoping in continuous faces includes several basic operations: undercutting,

breaking, loading or mucking of coal, its transport from the working face to the entry, timbering and stowing of the stope area, or caving of the roof in the mined-out space.

Most of the major operations in coal extraction are now fully mechanized.

Undercutting of coal. Breaking of coal should be preceded by undercutting with the aid of coal cutting machines.

Undercutting and breaking of coal with a miner's pick, which prevailed in pre-revolutionary Russia, was a hard and unproductive operation. Today it is not practised in the USSR.

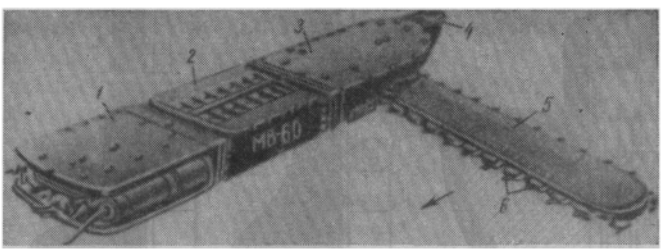


Fig. 80. Coal-cutting machine used at a stoping face:
1—cutter feed with drum, rollers and rope; *2*—electric motor; *3*—cutting subassembly with mechanisms actuating trim chain and gummer; *4*—gummer; *5*—bar; *6*—trim chain with cams and picks; arrow indicates direction of machine movement

The coal-cutter is designed to make a deep slit in a coal seam. Ordinarily, a lower cut is made, which facilitates the breaking of coal. This is done by shooting explosives placed in blast-holes.

The basic requirement for the coal-cutter is that it should quickly make an initial or opening slot (kerf) of sufficient height (12-14 cm), great depth (1.5-2 m) and thoroughly cleaned of gum.

In Soviet coal mines stoping is done with the aid of crossbar coal-cutters.

This machine (Fig. 80) has a long rectangular body at one end of which the operating mechanism—a bar, or jib—is fixed which during operation of the coal-cutter is at right angles to its body. The machine makes a cut, moving along the breast of the face; it has the following major subassemblies: the bar with the trim chain, the reducing gear of the bar drive, the gummer, the drum drive reducer, the electric motor with the master controller. The bar with the trim chain is the operating mechanism of the coal-cutter. It consists of a guide-frame along which the trim chain slides during operation. The trim chain is made up of connecting planks and cams to which steel picks with hard alloy inserts are fastened. The bar has suspended swing sockets and can be fixed at right angles to the machi-

ne body (during operation), or alongside it (when flitting the machine down).

The purpose of the gummer is to remove gum from the driving sprocket of the trim chain. The feed part of the machine sets it in motion. It is provided with a reducing gear rotating a drum around which a rope is wound (Fig. 81). One end of the rope is secured to the drum, the other to an anchor prop, or to the machine body. The coal-cutter travels by being pulled up towards the anchor post with

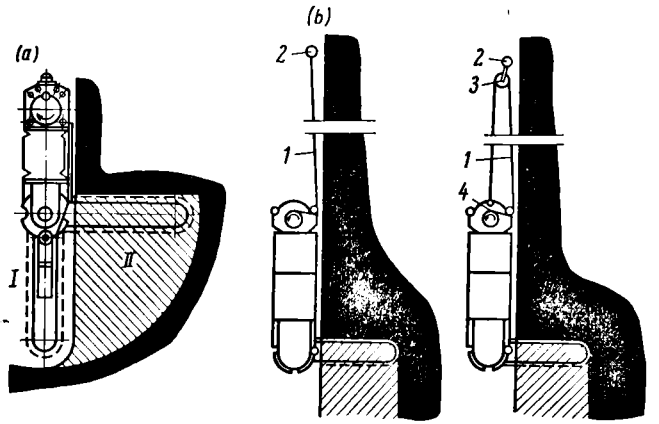


Fig. 81. Disposition of a coal-cutter and fastening of the feed rope:
a—disposition of machine in making initial cut; *I*—bar position at start of operation; *II*—bar position at end of operation; *b*—disposition of machine during operation and fastening of feed rope: on the left side rope is fixed directly to anchor post and on the right to cutter body; *1*—feed rope; *2*—anchor post; *3*—roller; *4*—cutter body

the aid of the rope. Apart from a low operational speed the coal-cutter can develop a flitting speed for a faster idle travel. The operator's stand is near the feed part of the machine, where all the control handles are concentrated; the handles are used for adjusting and shifting the speeds, switching on and off the cutting mechanism and the master controller. The electric motor actuates the reducing gears of the feed and cutting mechanisms of the machine.

The capacity of heavy-duty coal-cutters employed in the USSR reaches 15-20 thous. tons and more of undercut coal per month.

The position of a coal-cutter at the working face is seen in Fig. 81.

Breaking of coal. In breaking coal with the aid of explosives blast-holes are first drilled by hand-held or column-mounted electric augers.

Electric hand drills, weighing 22-28 kg, are fed from a 1-1.7 kW, 2,700 rpm electric motor operating from a 127 V mains. The power is supplied through a flexible cable.

Bits for these drills are either made of steel or have hard-alloy racks, or tips. Bits for drilling coal are manufactured from twisted drill steel made in the shape of single-point cutting tools with a deep flute between them. Column-mounted electric augers are used in drilling hard coal and also clayey and sandy shales.

To break coal by shooting explosives blast-holes are drilled to approximately the same depth as the cut. With a toe cut the holes are sited near the back, with a top cut near the bottom and with a centre cut near the bottom and the back. Blast-holes are spaced at 1-5 m, depending upon the hardness and structure of the bed. To reduce fragmentation of coal, no more than one or two ammonite sticks and in gassy mines the same number of permissible grisoutine cartridges are put into each hole. Explosives consumption per ton of broken coal is 15-40 g.

For ancillary work in breaking coal in sloping and steep seams pneumatic coal picks are used. Such a pick weighs 7.5-9.5 kg. It is made of steel, is 200-400 mm long, its point has a pyramidal shape. Electrically operated picks are also used for breaking coal in the mines of the USSR. They consume much less power than pneumatic picks.

Loading of coal. Formerly, coal was usually loaded onto a conveyer by hand. Today this operation is largely mechanized.

It is important to promote mechanization of the most laborious operations in coal mining and, above all, the mucking of coal at working faces and the loading of rock and coal when excavating development workings.

When driving headings in the coal mines of the USSR wide use is made of special combines, coal-loading machines with crab buckets and bucket-type rock loaders. In coal walls self-loaders are employed which are set up at a distance of 0.3-0.4 m from the breast of the face (to handle undercut coal after blasting). Mining combines of various design are also used.

The plans for the development of the Soviet coal industry envisage a further increase in mechanized mucking, the highest level to be reached in the mines of the Karaganda coal fields (95-100%).

Mining combines have been built on the basis of improved coal-cutters with curved and circular bars (Fig. 82).

Curved bars for coal-cutters were first employed in the USSR in 1931. The end portion of the bar, bent at right angles to the horizontal part, can dislodge coal from the breast of the face.

With a curved bar a special double-hinged chain bent in two planes is used.

Today "Donbass"-type mining combines are chiefly employed in the coal mines of the USSR in working gently sloping seams.

Below we present a brief description of the wide-range "Donbass" and narrow-range К-52м and КШ-1 mining combines. In the "Donbass" (Fig. 83) the operating part consists of a circular bar with a trim chain, and a rod with cutting disks.

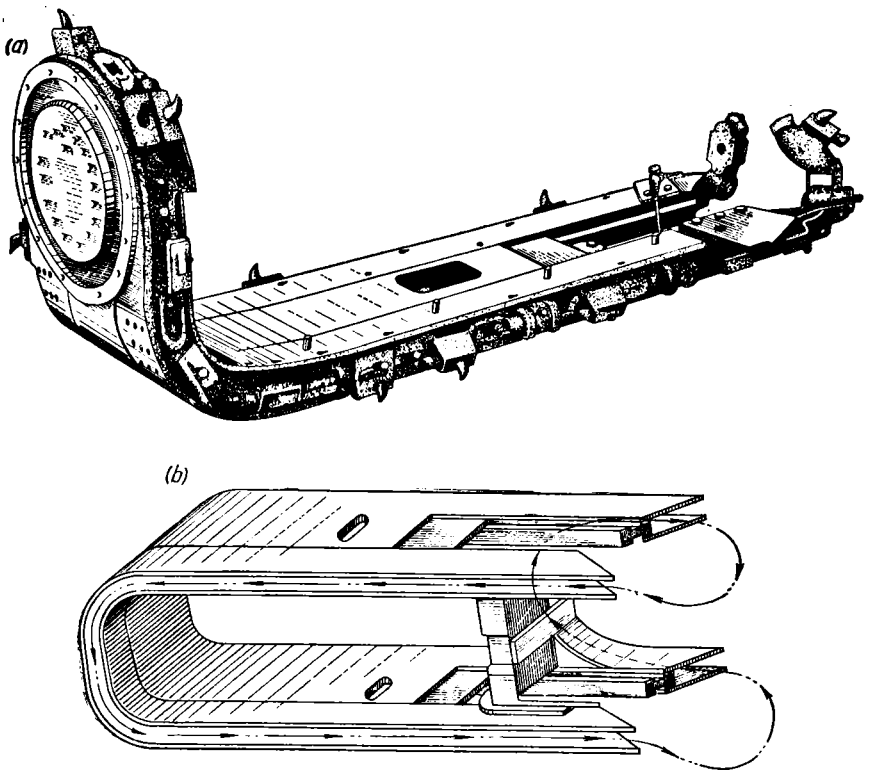


Fig. 82. A curved and a circular bar:
a—curved bar; *b*—circular bar

Coal, dislodged from the seam by the circular trim chain, is cut additionally by the disks and fragmented by the ballast rod. Behind the latter is a circular loader, whose chain is provided with special loading scrapers. The loader is powered by a separate 13 kW electric motor. The combine operator switches the loader on and off through the master controller.

When moving in the lower part of the machine the loader scrapers entrap the fragmented coal and transfer it onto a conveyer. The gum carried away by the trim chain of the circular bar is collected in the gummer housing and then loaded onto the face conveyer. The

length of the bar and rod, which determines the width of the coal strip extracted by the combine, can be varied from 1.4 to 2 m. The cutting speed is 1.9 m/sec. The main electric motor of the combine, which drives the mechanism of the operating part and shifts the machine during its operation, has a capacity of 65 kW. The feed rate may vary within the range of 0.27-0.54-0.81-1.08 m/min, depending upon the hardness of the coal. The flitting speed of the combine is 14.5 m/min. The overall dimensions of the machine are: length

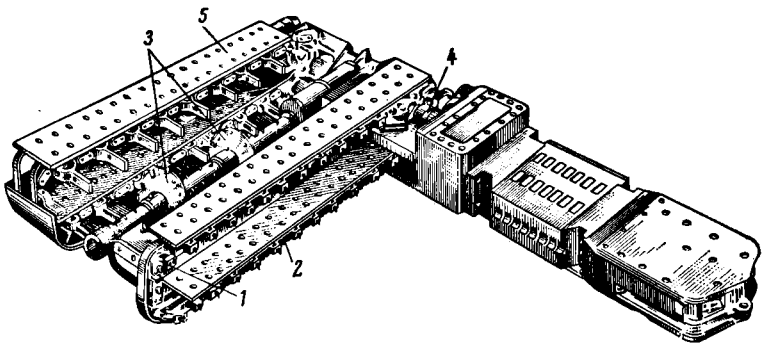


Fig. 83. Combination coal-cutting and loading machine "Donbass":
1—circular bar; *2*—toothed chain; *3*—rod with cutting disks; *4*—gummer; *5*—flight ring loader

4.41 m, width 0.72 m, height 0.7-1.6 m, depending upon the elevation of the circular bar.

In a number of mines coal-combine operators have achieved very high performance, the record output being 60,000 tons per month.

Alongside the application of the above-discussed coal miners with a working width of 1.45 m and more, in recent years narrow-range coal miners with a working width of 0.5-0.9 m have been introduced. Extraction of coal in narrow strips helps to cut down power consumed in fragmenting coal by squeezing it in a more rational manner; conveyers operate under better conditions, since they are accommodated in a post-free working face area, the feed rate rises and there are better conditions for supporting the roof in the coal wall, which facilitates the design of mobile supporting sets. The use of narrow-range sets with shuttle-type mining combines has raised the effectiveness of coal extraction both in long- and shortwalls (the rate of working face advance and the labour productivity of the facemen have increased considerably). As an example, Fig. 84 shows a set of equipment with a narrow-range К-52м combine. The set includes: a mobile face conveyer, metal props, breaker posts and hinged headboards. The conveyer is moved to a new roadway by using hydraulic

screw jacks linked with the conveyer frame and hydraulic anchor posts. The operating parts of the К-52м machine are two drums fitted out with toothed cams. The coal is loaded by the picks and cams of the operating parts, which throw it onto an inclined ramp set up near the operating mechanism.

The КШ-type continuous coal-mining machine (Fig. 69) is designed for a working range of 0.5-0.8 m and extracts coal from seams 1.5-2.5 m thick. It operates from a conveyer frame. The operating part of the machine consists of 2 screw conveyers with picks. The

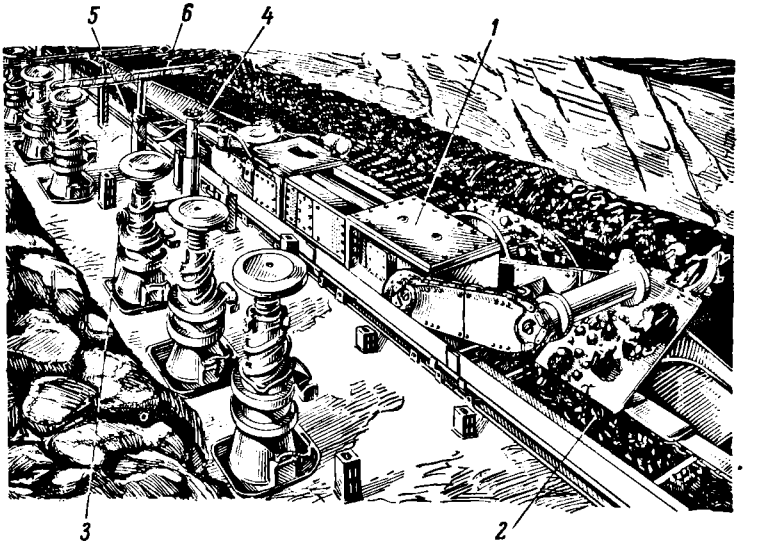


Fig. 84. Set of equipment with a short-range К-52м mining combine:
1—К-52м combine; *2*—mobile КС-9 flight conveyer; *3*—hydraulically operated breaker prop; *4*—hydraulically operated anchor prop; *5*—metal prop; *6*—articulated roof beam

screw conveyers can be fixed at different elevations with respect to each other, in accordance with the thickness of the seam. The cutting speed is 2.07 m/sec; the machine is equipped with hydraulic feed. The feed rate when running under load is 0-3 m/min and when flitting 0-10.5 m/min. Drawbar pull under load is 10 tons and when flitting 2.9 tons. The machine makes the first or initial cut at a feed rate of up to 0.02 m/min. The electric motor rating is 100 (75) kW. The machine weighs 10,642 kg. Its overall dimensions in the operating position are 8,500 × 940 × 1,220 mm.

The КШ combine is a shuttle-type machine and can operate in different directions, this being an important advantage. The operating potentialities of narrow-range continuous mining machines

employed as a part of a set together with mobile support and conveyers are very great. At the time of writing the record achieved with a KM-87 set was 110,050 tons of coal at one production face in 31 working days.

In the near future it is proposed to use in Soviet coal mines with gently sloping seams integrated coal-mining units allowing extraction of coal without permanent presence of miners at the working face. Thus, in the A-2 unit dislodgment of coal from the solid mass is effected by a high-speed toothed plough, or by a narrow-range continuous mining machine; coal transport is handled by a flight conveyer, timbering and roof control by self-advancing hydraulic supporting sets. The A-2 unit is controlled from a central desk located near the haulage entry, it is designed to produce 800 tons of coal daily (in two shifts), to secure an advance rate of 6-8 m in a working face 60 m long and labour efficiency of 20-22 tons per faceman per shift. Other, still more productive units have been built and are undergoing field tests (A-3, etc.).

In working beds of limited thickness some mines of the Donets coalfields employ coal planers. These are castings with coal-cutting knives inserted in them. The machines are pulled along the face breast by tugger hoists. They are provided with mouldboards for loading coal onto a conveyer. The planer strips the coal at a depth from 70 to 150 mm, depending upon its hardness.

At a planer-operated coal-face the set of powered equipment consists of a coal planer, all-in-one mobile flight conveyer and mechanisms for its shifting (Fig. 85). The planer is structurally integrated with a face conveyer, which serves it as a rigid guide and ensures its steady travel. More often than not a planer is pulled by a draw chain, whose driving mechanisms are also mounted on the conveyer frame. The travel mechanisms are usually pneumatic or hydraulic jacks spaced at 6-8 m at the coal wall.

The greatest number of planers are now used in extracting hard coal in the anthracite mines of the Donets basin, where the average monthly output per planer varies between 15 and 20 thousand tons of coal, with an average labour productivity per faceman of 225-250 tons per month. A special merit of the planers is their ability of mining very thin beds, where other machines cannot be employed except for combined drill-and-screw-conveyer units.

Transport. Without the mechanization of haulage the winning of coal with the use of coal-cutters, loading and continuous mining machines would be impossible.

Soviet coal mines make extensive use of flight or drag conveyers. Jigging conveyers and scrapers find no application today.

Belt conveyers are used in high-output production faces of considerable length. Their major elements are the driving mechanism

with an electric motor, the tension drum with a belt-lifting arrangement, the belt, and idle rollers. The above-enumerated conveyer parts are mounted on a frame made up of several sections.

Batch production of heavy-duty belt conveyers with wire-reinforced rubber belts designed for mining enterprises is underway in the USSR. Belt-conveyers with a high-resistance belt intended for longwalls of 800 tons/hr production capacity are being tested.

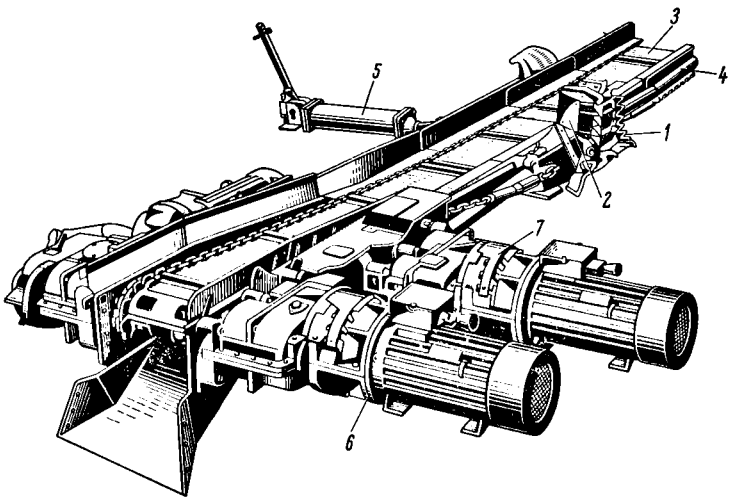


Fig. 85. General view of a coal planer:

1—plough teeth shearing coal chips off the working face; *2*—mining planer share for loading coal onto a face conveyer; *3*—face conveyer; *4*—planer guides connected to the conveyer frame; *5*—hydraulic jack to move conveyer together with planer to face breast; *6*—face conveyer drive; *7*—drive unit for propelling the planer

Experience shows that the severe conditions under which belt conveyers have to operate in coal walls often restrict their use.

A machine that successfully withstands these conditions and meets the high-output requirements is the flight or drag conveyer. It was first introduced in 1936. In subsequent years highly efficient flight conveyers were constructed, whose rigid structure permitted a coal-cutter to be mounted on their frame so as to make possible partial self-loading of coal (up to 35-40%) onto the conveyer. Extensive application of flight conveyers improved the organization of work in the production face and a continuous flow of coal to the haulageway. At a later date flight conveyers were adjusted so that narrow-range coal miners could move along their frame.

The basic parts of a flight conveyer (Fig. 86) are: the driving mechanism with an electric motor, sections of a stationary metal

trough, an endless chain carrying scrapers moving the coal along the trough, and the guiding drum.

New models of efficient flight conveyers have been designed and built in the USSR, for example the RC-9 flexible mobile conveyer for transporting coal in production faces of slightly dipping seams 0.8 and more metres thick. The capacity of this conveyer reaches 260 tons/hr, its horizontal length being 250 m (transport in a long-wall by a single conveyer). The transfer of conveyers is effected by

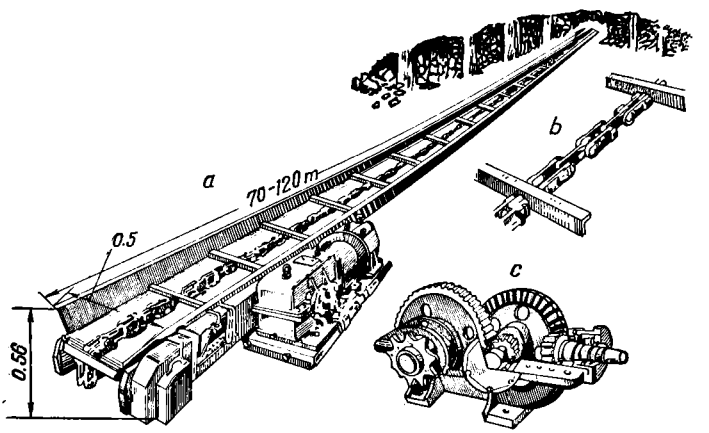


Fig. 86. Flight conveyer:
a—general view; *b*—chain link with flights; *c*—drive

metal support shifters or jacks. In recent years standardized flight conveyers with 2-3 chains and a production capacity of 270-580 tons/hr have been introduced in the USSR.

Flight conveyers of ordinary height are unsuitable for low seams 0.8 m thick. Therefore, Soviet plants, apart from standard flight conveyers, manufacture special ones with the idle and loader branches located in one plane. This design facilitates loading the coal onto the conveyer and enables the idle branch of the chain to be used for supply of timber to the working face. The greatest height of the conveyer (at the driving mechanism) is 430 mm. The efficiency of conveyer installations can be greatly improved by using remote control.

Timbering, artificial roof caving and filling of the mined-out space. As the roof of a seam becomes exposed the stope has to be supported. Timbering procedures and the types of support employed have been described in detail in the foregoing text. When the working face has advanced over a definite distance the roof is caved by the methods discussed before.

In mining a bed with partial stowing the mined-out space is filled with rock obtained by ripping the roof and bottom during the drivage of intermediate entries or special dummy roadways. The rock resulting from entry ripping is placed on both sides of each entry, or just on one of its sides on the dip (with the dip angle exceeding 6-8°). Complete filling is an exception rather than a rule in breast-stoping.

In working thin and medium-thick gently sloping and inclined seams one can do without complete filling, using induced caving or partial stowing and improved roof control methods.

Ventilation. In breast-stoping ventilation is a simple matter. The air current is passed from the haulage entry to production faces, which are swept by it in succession (Fig. 75). In mining gas-bearing beds the main air current is split into several subcurrents. These are passed through the gravity incline or its manway to individual stopes, one subcurrent being sent along the haulageway to the lower sublevel. The absence of dead faces in this system of mining favours proper aeration. Some complications in regard to ventilation may be encountered in arranging advance gravity inclines with dead headings. In this case local ventilation should be used.

The above fundamental principles of performing individual operations in breast-stoping are also applicable to the system of long-pillar mining along the strike and to many other methods, in particular to those employed in working thick beds by slicing.

Breast-Stoping in Steep Seams

As compared to flat-dipping beds the mining of steep seams is characterized by a greater hazard of caving and creeping of the floor, caving of coal in the stope and by the possibility of utilizing the weight of the coal for its transportation from the face. The presence of the caving hazard and of gas in the seam limits the level interval height, which usually does not exceed 120-150 m. A level interval height of up to 170-180 m is adopted in non-gassy mines and in those with low evolution of gases. A steep seam is mined starting from a holing chute driven parallel to a through-cut between the top and bottom entries (Fig. 87).

Like in mining gently sloping and inclined beds, breast-stoping is largely done with stable wall rocks. Depending upon the degree of wall rock stability mining may be done with and without division into sublevels. In the former case the coal from the upper sublevel is passed down via chutes spaced at 70-100 m (the total length of the conveyer set). In breast-stoping in steep seams coal is usually extracted without dividing the level into sublevels (Fig. 88*a*).

The coal is mined with pneumatic coal picks and continuous mining combines.

A feature characteristic of the mining methods employed in working steep seams with the coal broken by mechanical picks is the overhand shape of the stope (Fig. 88). Each bench accommodates one

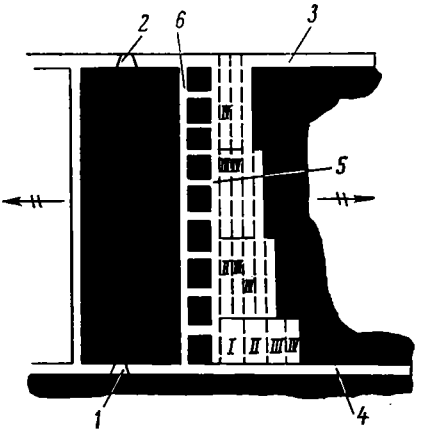


Fig. 87. General deployment of operations in continuous mining of a steep bed:
1—haulage crosscut; *2*—ventilation crosscut; *3*—top entry; *4*—bottom entry; *5*—holing chute; *6*—through-cut. *I*, *II*, *III*, *IV*—sequence of bench extraction

miner. The lower benches are always pushed forward in advance of the upper ones. The coal broken in the superjacent bench rolls down

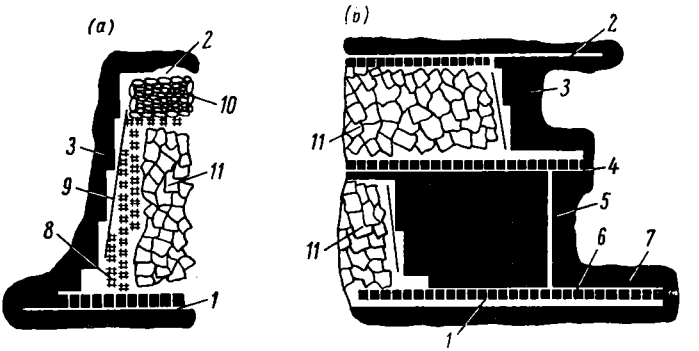


Fig. 88. Variants of continuous mining in extracting steeply pitching seams:
a—variant without division into sublevels and with partial filling; *b*—variant with division of level into sublevels and with excavation of dumping chutes (roof is caved into mined-out space); *1*—bottom haulageway; *2*—top entry; *3*—bench; *4*—sub-entry; *5*—slope; *6*—through-cut; *7*—back entry; *8*—crib; *9*—vibroconveyer; *10*—mine-fill; *11*—caved rock

along the dip, bypassing the underlying benches, and in this way the miners are always protected by the solid mass of coal.

In mining steep seams by breast-stoping, development proceeds with a slight advance over stoping operations and consists of driving

the top, bottom and intermediate entries in coal with partial ripping of the rock within the sectional area of the entry. Subentries are driven when a level is divided into sublevels.

The ripped rock is used for stowing, or else it is laid down by the side of the entry (rib-fills or pockets), the corresponding amount of coal being extracted previously.

Stoping in a level starts from a holing chute. The adopted bench advance is between 1.5 and 3 m. Owing to the progressive mining practices the height of the benches has been substantially increased and now reaches 20-40 m (depending upon the hardness of the coal and the rigidity of the wall rock), compared to the former height of

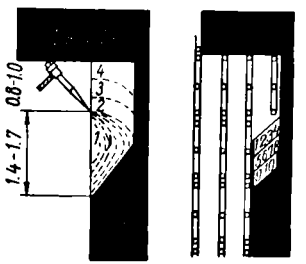


Fig. 89. Cutting top corner and sequence of breaking coal in a bench

4.85-8.5 m used when coal was broken by hand. The length of the bench is ordinarily fixed in such a way as to allow the faceman to break off a coal strip of width equalling that of the timbering and length equal to that of the bench. The coal in the bench is broken off with a pneumatic coal pick, commencing from the upper part of the bench (top corner, or starting slot) and proceeding towards the lower part. In drawing coal each faceman stands on a propped up working platform.

Details of coal extraction in a bench are shown in Fig. 89. Methods of operating a coal pick depend upon the structure and physico-mechanical properties of the bed. Habitually, the cutting of the top corner is started 0.8-1 m below the face toe. After this portion of coal has been cut over the width of the timbering (0.8-1 m) the top bands of coal (2, 3 and 4) are broken down, following which the coal is broken off in the remaining part of the bench.

In an overhand stope (Fig. 90) timbering is arranged in regular rows on the dip and along the strike. The props in the row are spaced at about 1 m. With weak wall rocks wood slabs are inserted between the props and the floor and roof, placed end-to-end, and additional lagging of the floor and roof is done. Cribs are often set up in the stope area in addition to the face timbering props. The cribs are laid down on special posts, or rest upon ordinary timbering (Fig. 90).

Spacing between the cribs depends upon the firmness of the rocks. Along the strike the rows of cribs are usually spaced at 3-6 m, and the individual cribs in a row at 2-4 m.

With a steep angle of dip props are not knocked down because of the hazards this entails, while cribs are transferred to another place under favourable conditions only. The coal broken off in the stope falls down by gravity. Troughs are set up to direct its flow along the face. The troughs are made of wood boards and nailed to the

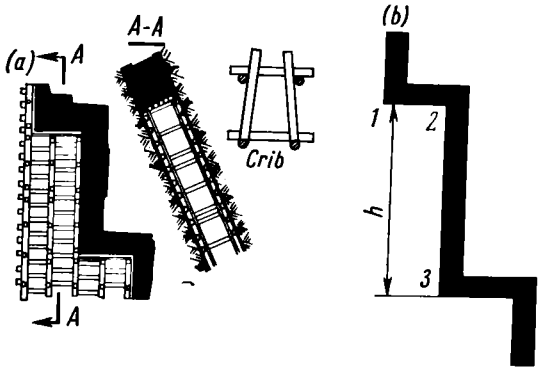


Fig. 90. Disposition of timbering and bench elements:
a—timbering; *b*—bench elements: *1-2*—bench overlap; *2-3*—bench height; *2*—top corner of bench

props. When there is a band or intercalation of dead rock in the coal bed, which has to be separated directly at the face, the rock is thrown into the stoped-out space; to do this, clearances are left between the individual troughs. The troughs are terminated at some distance from the lower part of the level, so as to provide a space for some reserve of coal. If sill pillars are left over the haulageway the latter is pierced with crosscuts at definite intervals. From the working face the coal is passed to the crosscut, whence it is loaded into mine-cars through a dumping chute. The minimal distance from the bench to the troughs is 2-3 m. As the benches advance the troughs are shifted forward every 4-5 m.

Steep seams are extracted either with caving of the wall rocks or filling of the worked-out space (Fig. 88). The top entry is supported by a packing strip, and for this purpose a special flooring is made in the open goaf on which barren rock is placed. This flooring rests on timbering set up earlier and reinforced by additional props and one or two rows of cribs. The sources for the mine-fill are: interlayers of barren country rock in the seam, rock obtained by slashing the bottom and back of the bed (false bottom, false roof), by the ripping

of entries, by the drivage of rock headings, and also stowing material specially delivered from the ground surface.

In working with sublevels the upper sublevel should be pushed forward in advance of the others.

Now, let us consider some variants of breast-stoping in steeply pitching beds.

Breast-stoping with a continuous face throughout the level. This variant was used in undercutting coal with a coal-cutting machine held in the face by a safety rope, with one of its ends secured to the machine and the other to the tugger hoist put up in the ventilating entry.

Because of the difficult conditions of operating coal-cutters in steep seams mechanical undercutting of coal did not find extensive application; therefore the pneumatic coal pick had long been the basic tool for the purpose. In 1950 a KKП-1 continuous coal miner for operation in steep beds was built (Fig. 91). In a continuous face it operates downhill along the dip of the bed and draws coal from an oblique bench, taking off a strip of coal from 2 to 2.3 m wide over the entire length of the coal wall. The machine is suspended on a cable fixed to the drum of a tugger hoist, which is put up in the top level. Coal in the face is dislodged by the reciprocating movement of a breaking device (with the operating member in the shape of a rotating bit) along the frame of the machine. The general view of a coal wall with a KKП-1 machine operating in it is shown in Fig. 91.

The bit shears coal off from the breast of an open face in layers from 50 to 200 mm deep (across the working face), depending upon the hardness of the coal. When sheared off the coal rolls down along the slope of the face. The continuous coal miner is adopted to operate in seams of variable thickness (0.75-1.3 m). During operation the machine rests with the three bearing plates of its main frame upon the foot wall. Blades in the bearing plates ensure the required orientation of the machine down the dip. When working in hard coal a special skid is mounted additionally to the bearing plates, which slides along the row of props next to the face and prevents the machine from going off its course within the face area. The continuous coal miner is equipped with a pneumatic engine. Face timbering is put up from the top down the dip in the wake of the advancing machine. Under normal conditions the machine travels at a speed of 6-7 m/hr.

Now the KKП-1 continuous miner has been replaced by the KKП-2 model (Fig. 92), which is chiefly used in mining coal of different hardness in seams 0.8-1.3 m thick dipping at 45-75°. The design features and operation of the KKП-2 machine are similar to those of the KKП-1 model, the only difference being that the KKП-2 machine is fitted with two cutting bits. The width of the continuous miner in the transporting position (when pulling it up) is only 800 mm,

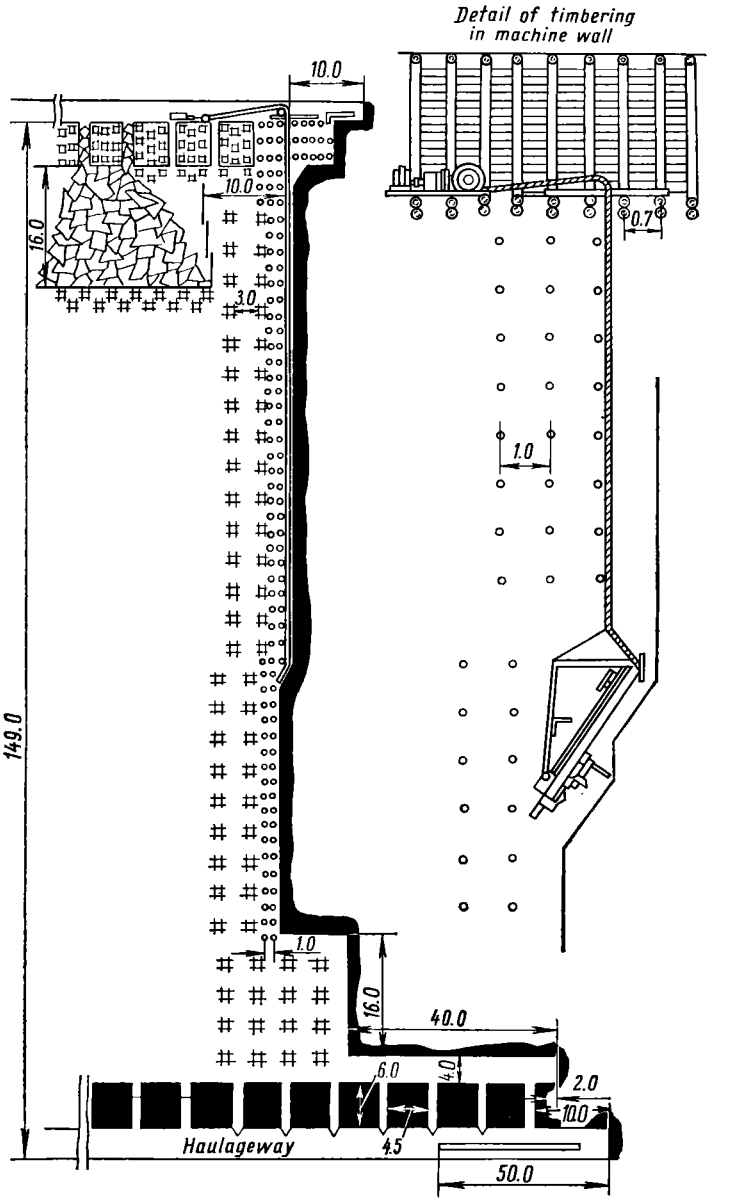


Fig. 91. Operation of ККП-1 continuous coal miner in a face of a heavily pitching bed

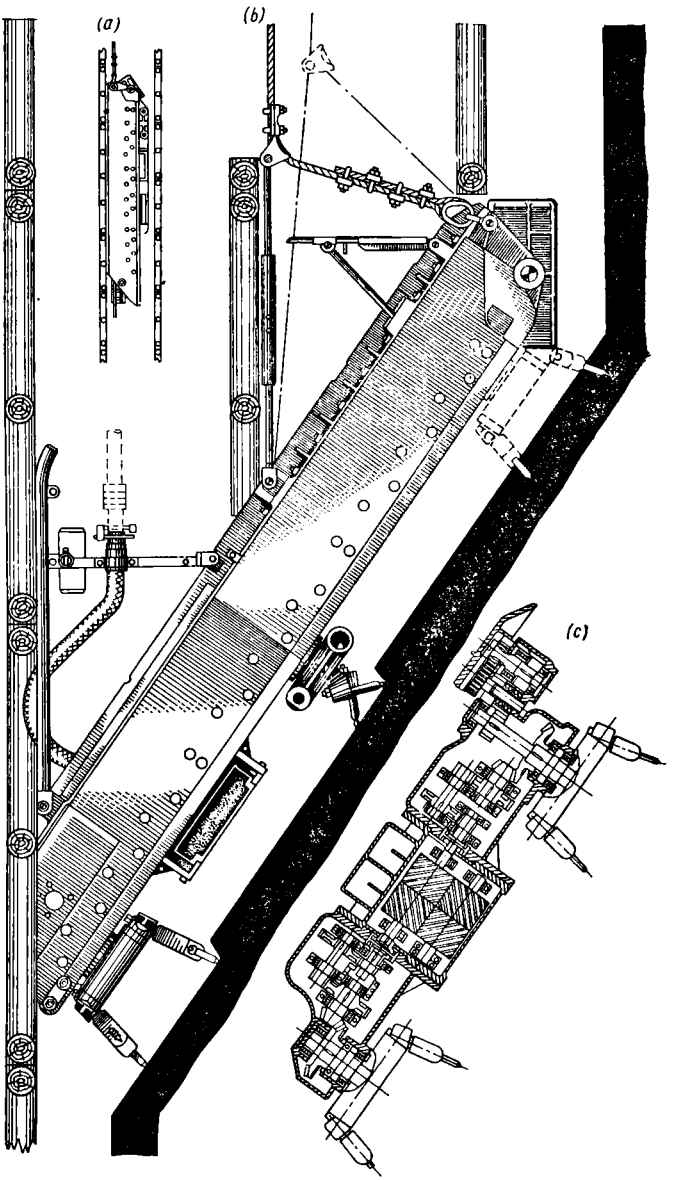


Fig. 92. KKП-2 coal-mining combine:
a—position in flitting; *b*—general view; *c*—kinematic diagram of operating mechanism drive

which enables it to travel between rows of timbering props. The production capacity of the ККП-2 machine varies between 50 and 190 tons/hr, depending upon the thickness of the seam.

Of late, use has been made of К-19 mining combines for working seams 0.45-0.80 m thick, and of К-55 combines for mining 0.7-1.3 m beds. The latter machine is provided with a narrow-range drum-type operating mechanism. Two cutter heads are fitted out with

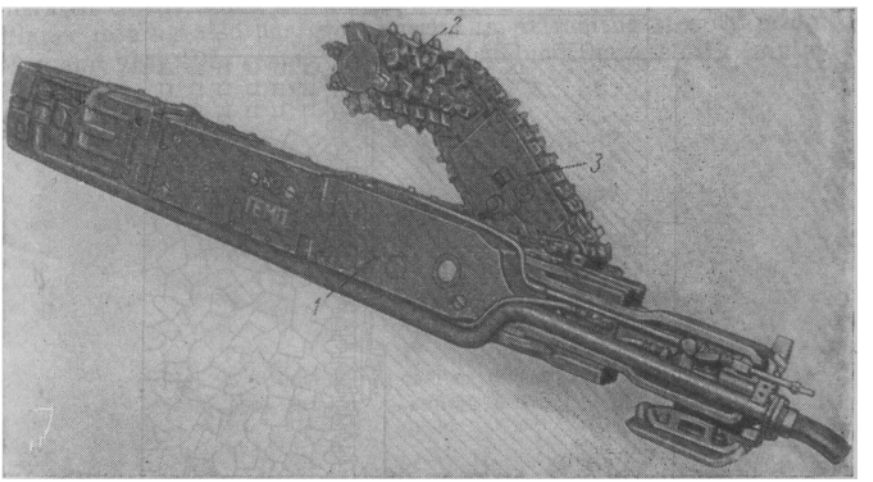


Fig. 93. Narrow-range continuous coal miner of the УКР type:
1—machine body; *2*—cutter head with picks; *3*—flat chained cutter jib

cams carrying cutting picks. The К-55 machine operates downhill. Among the latest designs that have gained wide recognition mention should be made of the narrow-range УКР model with a working range of 0.9 m. The machine is intended for extraction of coal of any hardness in seams 0.6-1.1 m thick dipping at 45-80°. It operates uphill in a bench of a continuous face, which is timbered when the extraction of a coal strip over the entire length of the coal wall has been accomplished.

The drum-type operating mechanism combined with a trim chain adapts itself to the thickness of the bed (dislodges and fragments coal in the seam throughout its entire height). The breast of the face is shaped with a flat jib. The leading skid, cast integral with the body of the cutting member, ensures stability of the machine in the face (Fig. 93). The continuous-miner operator controls the machine and the hoist at a distance of 10-15 m uphill from it, by

using a portable control panel. The working range of the УКР continuous miner is 0.9 m, its output reaching 50-80 tons/hr. The record production with the use of a УКР machine is 65,000 tons per month.

In recent years large-scale trials have been conducted on hydraulically powered mine-support sets of the "Dnepr" type (Fig. 94) intended for mechanization of face timbering and for roof control

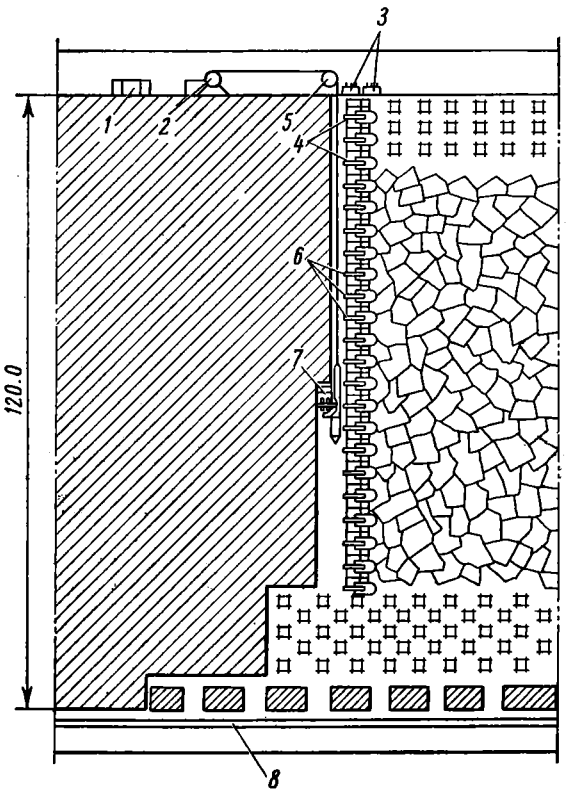


Fig. 94. Disposition of УКР continuous coal miner combined with mobile "Dnepr"-type support set:

1—oil pumping station; *2*—tugger hoist; *3*—control panels for support set; *4*—anchorage; *5*—roller; *6*—support frames; *7*—continuous coal miner; *8*—conveyer

by complete induced caving in stable and medium-stable wall rocks of seams 0.9-1.3 m thick and integrated with an УКР machine. Among the advantages of the "Dnepr" mine-support set are: simplicity of its design, remote control from a ventilating entry, complete

head-lagging, high rate of advance, and the possibility of finding simple methods for automation of its operation. The use of the "Dnepr" mine-support set has cut timber consumption from 65 to 12 cu m per 1,000 tons of coal yield. The field of application of mining combines in working steep seams is limited mainly to beds with stable wall rocks. In addition to new combines coal planers are being introduced for operation in the mines.

Advantages and future prospects of planer mining have been enumerated earlier when discussing the working of gently sloping beds. Planer mining also has great merits in extracting steeply pitching thin and very thin seams. Of the coal planers successfully employed

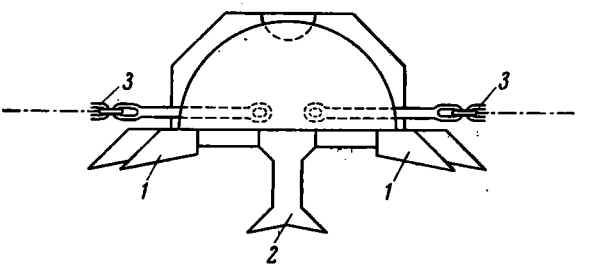


Fig. 95. Ram-plough coal extraction mechanism:
1—side shearer; *2*—central shearer; *3*—chains

in mining steeply dipping beds mention should be made of ram-ploughs hauled along the production face by means of a pulling chain, which withstands a breaking force of 30-50 tons. The operating member of the ram-plough (Fig. 95) is made as a sectional or all-cast structure with massive hard-metal tipped side shearers fixed at its ends. In the middle of the ram-plough is a central shearer placed somewhat in front of the side ones. A driving motor of 40-50 kW capacity ensures a travel speed of 1.2-2 m/sec for a plough moving along the working face. The central shearer penetrates the face and shears the coal off concurrently with the side shearers. Shuttle-type ram-ploughs usually operate at an angle of 65-75° in a 100-150-m long coal wall.

Breast-stoping with an advanced top sublevel and dumping chutes made in the solid mass of the bottom sublevel (Fig. 88). Dumping chutes are made every 70-100 m. The advantages of this variant include greater safety and complete independence of operations in individual sublevels. The drawbacks are: the necessity of additional development work (dumping chutes, intermediate entries) and the hazards involved in running bottom sublevel benches across dumping chutes.

General Conclusion on Breast-Stoping

The relative proportion of breast-stoping should decrease considerably in the next few years. This is due to a number of disadvantages inherent in advance working of a mine-field by this method.

Now that entry-driving machines and equipment ensuring high-speed excavation of mine workings are built and widely used, it is possible to change over from advance working of mine-fields and producing sections to retreat mining, which is more effective and technically expedient. Under such conditions long-pillar methods of mining are more rational.

4-7. PILLAR METHODS OF MINING

General

Pillar mining is employed chiefly in seams of medium thickness with low-rigidity wall rocks. The working of such seams is attended by intensive roof caving. Under such conditions it is technically difficult to maintain workings in the mined-out area, and for this reason in pillar methods the working section of mine openings is always located in an intact solid mass of coal, the undermined sections of the workings being gobbed up.

As distinct from breast-stoping, in the pillar method prior to extraction the solid mass of the mineral in place is broken by development openings into sections, so-called pillars. This method is certain to gain wide acceptance.

The most common of the pillar methods is mining by long pillars on strike. Mining with long pillars up the dip, with diagonal and short pillars is not so widespread. In mining flat and gently pitching seams variants of the long-pillar method—with double or single walls are employed (the Moscow coal basin). We will now give a more detailed description of long-pillar mining on strike and a brief review of other pillar systems.

Long-Pillar Mining on Strike in Slightly Sloping and Inclined Beds

Depending upon the method of mine-field development, the dip angle of the seams, their thickness and gas abundance, several modifications of long-pillar mining on strike are employed, namely: with and without division of the level into sublevels, with coal delivery to the front or rear gravity incline, with one-way or two-way mining, with through or separate ventilation of the production faces (in gassy mines).

Let us consider this method with special reference to the level-wise mode of mine-field development with division of the level into sublevels.

Development work. In this method (Figs. 96, 97) a level is broken up into extraction panels, which are serviced by their own gravity

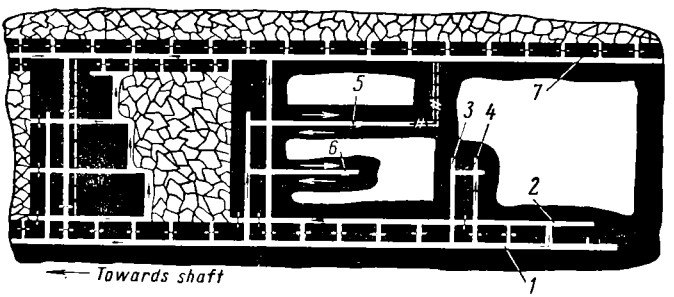


Fig. 96. Long-pillar mining on strike with pillar recovered in the direction towards rear brake incline:
1—haulageway; *2*—ventilation back entry; *3*—brake incline; *4*—manway; *5* and *6*—sub-entries; *7*—ventilating entry

inclines or dump chutes, depending upon the dip angle. The extraction panels are consecutively prepared and stoped by the full-advance method, or, more often, in the reverse order. Gravity inclines are made by driving narrow or wide headings with a side or gate

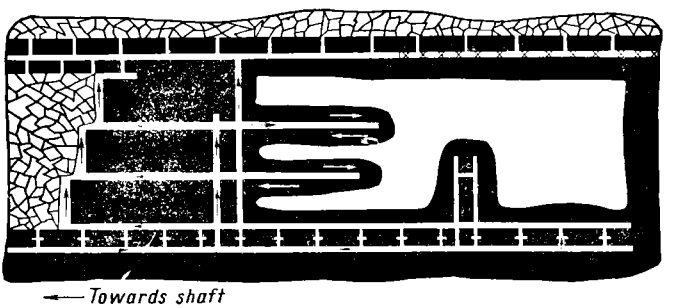


Fig. 97. Long-pillar mining on the strike with pillars recovered in the direction towards front gravity incline

pack. In an inclined bed dump chutes are made by driving narrow headings. Sub-entries are pushed forward starting from the gravity incline. The gravity manway is connected with the ventilating entry. In long-pillar mining the level is usually worked out through individual sublevels, the inclined height of the sublevel interval

being usually within the range of 80-150 m, depending upon the mode of mining and stability of wall rocks. The inclined height of the level interval is 300-500 m (two-three sublevels) depending upon the thickness of the bed, the presence of gas, the adopted mechanization pattern and organizational setup.

Level entries are driven with a down-dip side pack (Fig. 98.) or as narrow headings. To protect the haulage entry, sill pillars of coal are formed by driving a special back entry parallel to it without ripping the wall rocks. The haulage and back entries are connected by crosscuts

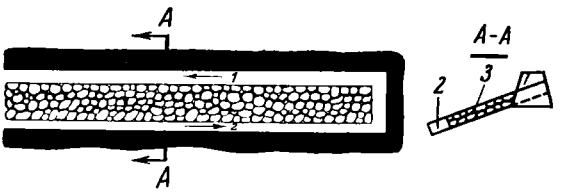


Fig. 98. Driftage of an entry with a down-dip packed goaf (plan view):
1—entry; *2*—air gate in packed goaf; *3*—goaf gobbed up with barren rock obtained by slushing the entry; arrow indicates direction of air flow

In relatively high seams construction of packing strips near the level entry instead of leaving sill pillars is not practised in pillar mining.

Among the advantages of coal delivery to the front gravity incline are the possibility of utilizing one gravity incline for deployment of both stoping and development work in one wing of the level, the absence of holing chutes in the extraction panel, and the smaller losses of coal in pillars at the gravity incline.

In the foregoing text mention has been made of the basic principles of mining with unilateral gravity inclines. In mining practice the long-pillar method is also employed with bilateral gravity inclines (Fig. 99). In this case gravity inclines are driven not near the boundary, but in the middle of the panel serviced by them. With bilateral gravity inclines intermediate entries are driven starting from the gravity incline towards the boundaries of the extraction panel, whereas stoping proceeds in the opposite direction.

In pillar mining there must always exist a definite correlation between stoping and development work, viz. a new gravity incline panel should be fully prepared for winning by the time it is to be opened for stoping operations. To ensure uninterrupted mining, development work should be done 3-4 months in advance.

Stoping operations. After development openings have been made long pillars on strike are formed in the extraction panel. Individual pillars are then recovered by advancing upper sublevels and haulage of coal to the gravity incline (Figs. 96, 97).

The sublevel advance rate is set at 10-40 m, and to assure a greater firmness of the roof of the front line of the working face is not made continuous, but broken. The advanced stoping of upper sublevels allows one to use the panel gravity incline, protecting it with coal pillars. The pillars can be extracted by a continuous face, or by butt-entries. Coal is usually mined out with a continuous straight face following the line of hade, irrespective of the cleat. All major mining operations involved in coal winning, loading and haulage to the rail track, timbering of the goaf area, roof control and organization of work are similar to those used in breast-stoping.

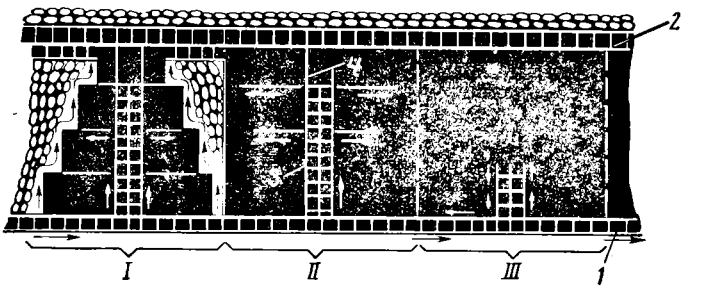


Fig. 99. Long-pillar mining on the strike with bilateral gravity inclines: *1*—bottom haulageway; *2*—top entry; *3*—gravity incline; *4*—manway; *I*, *II* and *III*—gravity incline panels; arrow indicates direction of air current (ventilation stopings are not shown)

In retreat mining of a level the protective pillars at the ventilating entry may be robbed as the face front line moves towards the shafts (with central location of the shafts). The retreat working of a level has also other advantages, which have already been discussed in detail. It should also be borne in mind that when the general direction of mining from the mine-field boundaries is towards the shaft there is less strain on the transport of coal to the front gravity incline than when it is brought to the rear incline.

Ventilation. In long-pillar mining the ventilation scheme is rather complex, and this is a substantial drawback of the system. Aeration is realized in the following manner: the main air current flowing along the haulageway is split into a number of subcurrents. One of them is passed to the active gravity incline, another directly to the working face of the lower sublevel, and a third to advance headings of entries, gravity incline and manway. The air current passing through the active gravity incline joins on its way the current flowing from the haulageway directly to the working face. The greatest difficulties in ventilation are encountered in advance headings of a new gravity incline, where it often becomes necessary to use auxiliary fans for local ventilation.

Long-Pillar Mining in Steep Seams

In working steep seams several modifications of the pillar method are employed: long pillars on strike with and without division into levels and sublevels, long pillars on strike with recovery by strip mining up the dip, temporary shrinkage of coal and extraction of pillars in strips down the dip. Here we will give a brief description of a frequently used modification of the system of long-pillar mining on strike with division of a level into sublevels (Fig. 100).

The conditions for the use of this system are common to all pillar methods. To prepare a level for extraction, level entries are driven

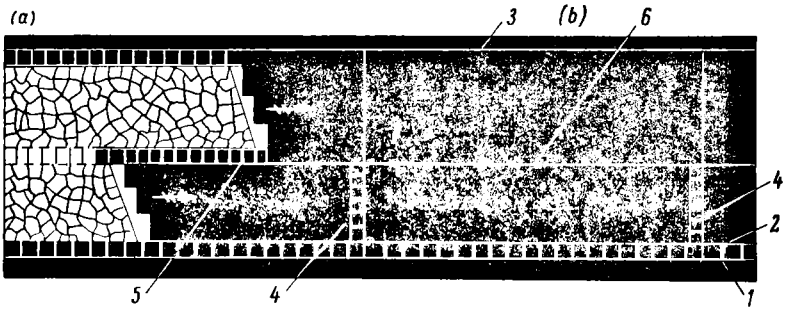


Fig. 100. Overhand long-pillar mining on strike:
a—chute section being extracted; *b*—chute section in development; *1*—haulage entry; *2*—ventilating through-cut; *3*—ventilating entry; *4*—operating chute with ladderway; *5*—sublevel entry; arrow indicates direction of stope advance

and connected by dumping chutes or slopes spaced at 100-300 m. One or two intermediate entries are pushed forward from the slopes, the number of entries depending upon the inclined length of the level interval (60-150 m) and the stability of the wall rocks. To protect the entries from caving, sill pillars are left measuring 4-6 m on strike and 5-10 m on raise. When mining at great depth the size of the pillars is increased to 20 m. Slopes and entries are of narrow section. Actual mining is done similarly to breast-stoping. The method of breaking determines the shape of the working face, which is overhand in mining with pneumatic coal picks and blasting, and straight when using coal-mining combines. As the coal is drawn the host rocks enclosing the seam cave in. In working coal beds susceptible to self-ignition the mined-out space should be filled. From the upper levels the coal is hauled along the corresponding intermediate entries to the dumping chutes sited somewhat farther from the production faces. In the intermediate entries coal is ordinarily transported by gate conveyers.

Variants of Long-Pillar Mining in Flat Beds

Mining practice knows variants of the system with the use of single and twin faces (walls). These are employed in working flat beds (lenses) in the Moscow coal basin under the following conditions of mining: a limited size of sheet deposits (length 1-1.5 km, width 0.6-1.5 km), lenses 1-3 m thick on the average, an undulating bottom; poor and unstable, often aquiferous, enclosing rocks, coal of low rigidity.

Because of the irregular outlines of the coal lenses and the nonuniform occurrence of the bottom the main entries are driven in different directions from the shaft. They break up the mine-field into separate parts. Each of these is divided into panels by face entries. Panels, in turn, are divided into long pillars by panel entries.

When choosing the direction of the main entries and panelling it is necessary to take into consideration the conditions of securing run-off of mine water to the shaft and normal gradients for the mine transport.

In the Moscow coal basin the variant of mining with twin coal walls (faces) is preferred. In this case two adjacent coal walls are joined both technically and organizationally and have one common mother entry (Fig. 101*a*).

The parameters of the twin-face variant of the system are: the length of the pillar, the distance between the ventilating breakthroughs, and the width of the chain pillars.

The choice of the length of the pillar and the coal wall is largely determined by the production capacity of the machines employed in mining and by the conditions of maintenance of the workings in unstable host country rocks. In the mines of the Moscow coal basin the coal is transported along the working face by flight conveyers, and along the mother entry by gate belt conveyers and flight conveyers. The conditions of maintenance of workings allow the length of each coal wall to be set at 50-80 m and that of a pillar 500-600 m. The adopted distance between the ventilating break-throughs driven in a pillar is 100-200 m and the width of the chain pillar protecting the entries, 6-12 m. Long pillars are mined starting from the boundary of the extraction panel towards the face entry.

In drawing coal from each of the working faces the general pattern of mining is the same as in the long-pillar method on strike.

In sections containing geological disturbances and faults and involving particularly difficult hydrogeological conditions of mining the single-face extraction variant is used (Fig. 101*b*). With this variant a single pillar is mined. Each pillar is cut up by two butt entries and then worked out from the boundary line towards the

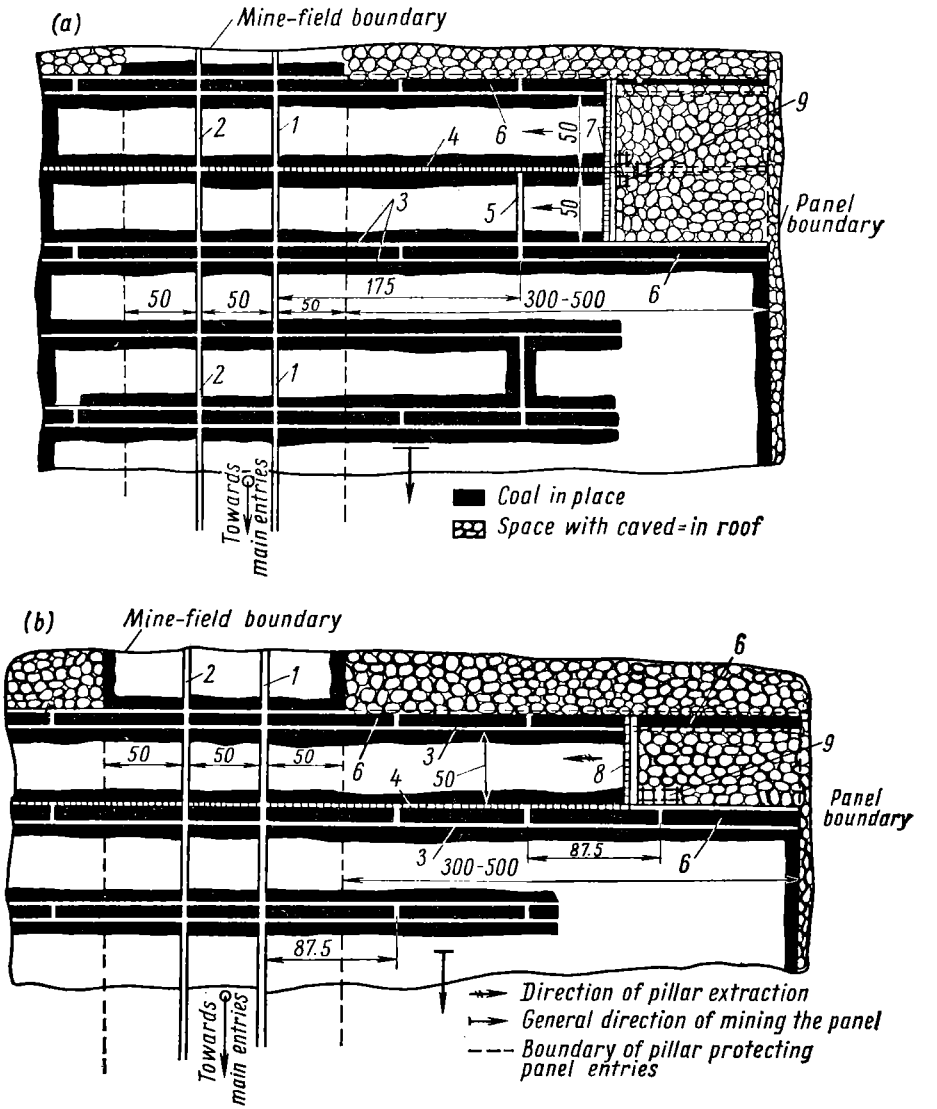


Fig. 101. Long-pillar mining of a panel:

a—twin-face mining variant; *b*—single-face mining variant; *1*—panel haulageway; *2*—panel ventilating entry; *3*—butt entry; *4*—butt entry with installed conveyer; *5*—ventilating through-cut; *6*—intervening pillar; *7*—twin face; *8*—single face; *9*—cribs protecting terminal end of conveyer

panel entry. A gate conveyer is installed in one of the butt entries. Concurrently with the drawing of coal the chain pillar between the working faces is partly robbed.

Pillar-and-Stall Method of Mining

The pillar-and-stall (bord) method of mining finds only limited application since it has substantial shortcomings as compared to long-pillar mining. These include the sizable amount of preliminary and subsidiary development work; the scattered pattern of production faces; the great number of openings in the solid mass of coal to be mined, the difficulties attending the mechanization of work, the high coal losses.

Pillar-and-stall mining is partly employed in the Moscow coal basin in extracting marginal sections of irregularly shaped lenses (Fig. 102), working-out of long-pillar portions (sill and

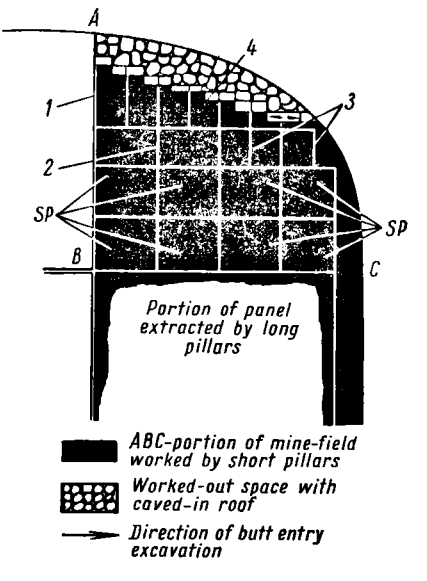


Fig. 102. Extraction of a pinched-out portion of a seam by the pillar-and-stall method:

SP—short pillars to be extracted; *1*—panel entry; *2*—temporary workings dividing solid coal in place into short pillars; *3*—working dividing short pillar butt entries; *4*—extracted through pockets

intermediate pillars) pierced by cut-throughs, robbing of long pillars when there are numerous quick-sand inrushes onto butt entries, to by-pass the sites of such inrushes.

In pillar-and-stall mining the solid mass to be mined is first cut up into pillars measuring 20 × 20 and 25 × 25 m. The short pillars are then separated by entries into individual strips 5-8 m long, each one of which is extracted through pockets. A pocket is an opening 2.5-3 m wide and 4-5 m long. The coal from the pocket is drawn in the direction from the strip entry or a subsidiary development opening until its boundary line has been reached. As the coal is extracted the pocket is timbered immediately. The coal in the pocket is broken by pneumatic coal picks and trammed by hand in mine-cars to the main haulageway. Once the coal is extra-

cted from it the pocket is caved by pulling out the timbering pieces. A new pocket is made near the worked-out one (leaving a small pillar in-between) or adjoining the previously mined-out pocket.

4-8. ROOM MINING

In full-seam room mining long parallel workings (rooms) 5-10 m wide are excavated with a permanent coal pillar abandoned between them, whose width is habitually smaller than that of the room (Fig. 103).

Room mining is practised chiefly in working flat and gently sloping beds. In mining steep seams the coal in the room is allowed to shrink, i.e. it is piled up provisionally and then drawn out.

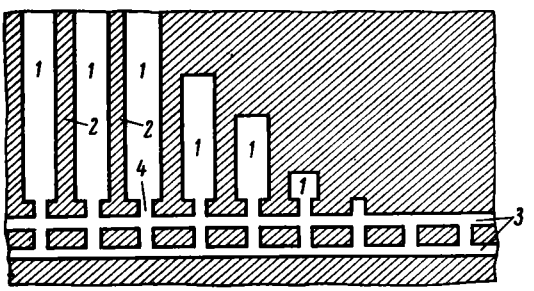


Fig. 103. Room mining:
1—room; *2*—pillar; *3*—entry; *4*—manway to room

In room mining coal losses are 25-30 per cent or more. This system is used extensively in Canada and the United States. In the USSR and West European countries it finds very limited application.

The preparation of mine-fields, the driftage of development openings and the pattern of stoping operations in rooms are similar to those used in the room-and-pillar method described below. Losses in the room-and-pillar method can be substantially reduced as compared with the room system.

4-9. COMBINED METHODS OF MINING

In mining low and medium-thick seams by combined methods use is made of the principles forming the backbone of the breast-stoping and pillar systems (for example, mining with twin entries and the room-and-pillar method).

Twin-Entry Method of Mining

In mining with twin entries (Fig. 104) the mine-field is divided into bilateral gravity incline sections. Each of these is, in turn, subdivided into sublevels, part of which are worked out by breast-stoping from the gravity incline towards the boundary of the section, while the remaining sublevels are extracted by applying pillar methods in the direction from the section boundary towards the gravity incline. The system owes its name to the twin entries driven in sublevels during extraction of coal by breast-stoping. The roof

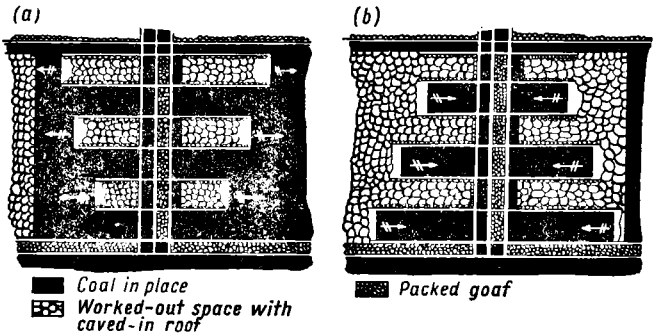


Fig. 104. Twin-entry mining:
a—coal section to be taken out first by breast-stoping starting from gravity incline; *b*—coal section to be extracted in second mining by long pillars starting from field boundary towards gravity incline; arrows indicate direction of stoping

rock slashed in driving the entries is laid down near them. The amount of rock should be sufficient for constructing packing strips. The method is employed with dip angles of maximum 10-15°, bed thicknesses within the range of 0.8-1.2 m, soft wall rocks and limited methane evolution.

Under these conditions of mining it is difficult to use the ordinary breast-stoping or pillar methods.

The major advantages of the method are: the reduced volume of development work (intermediate entries are formed in the course of stoping operations) and the safety of mining, since in breast-stoping the faces are protected by intact coal pillars in the neighbouring sublevels. With twin-entry mining it is easy to arrange separate ventilation of active working faces in all stages of coal extraction.

Room-and-Pillar Method of Mining

The room-and-pillar method (Fig. 105) is widely used in coal mines of the USA in working flat and gently sloping beds 1.5-2.5 m thick with very firm back and bottom of the seam. In the USA abou-

40 per cent of the entire underground coal yield is obtained by this method.

The mine-field is broken up by entries into a number of extraction panels, each 750-1,500 m long. Face entries are often run in groups of two-five entries in a panel. Each of the entries serves a special purpose: haulage, passage of men, air supply, piling of rock broken

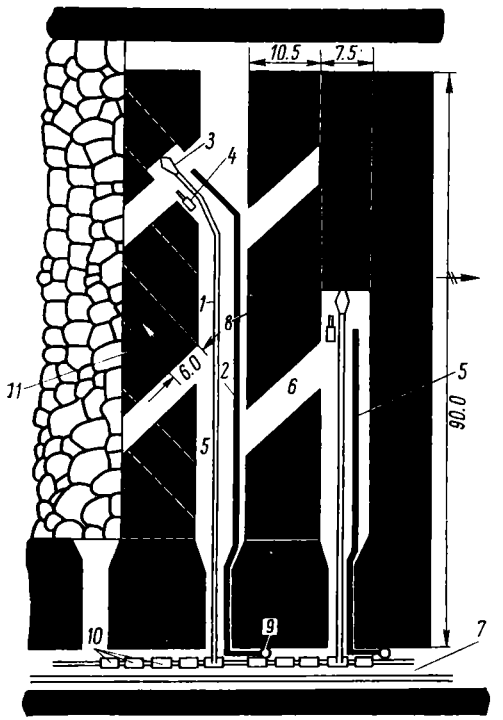


Fig. 105. Combined (room-and-pillar) method of mining. Conveyer transport version (timber props not shown):

1—conveyer; *2*—ventilation tube; *3*—loader; *4*—blast-hole drilling machine; *5*—room; *6*—through-cut; *7*—panel entry; *8*—pillar; *9*—fan; *10*—mine-cars; *11*—boundary line of recoverable pillar sections; black arrow shows general direction of stoping in panel; white arrow, that in pillar sections

by slashing and ripping, etc. Rooms are excavated from panel entries; the width of the rooms varies from 6 to 12 m, depending upon the thickness and stability of the wall rocks. The rooms are separated by rib pillars of the same or larger size. The length of the rooms corresponds to the distance between the panel entries (70-100 m). In recent years the length of the rooms has reached 300 m due to continuous mining of rooms and the use of extensible conveyers. The rooms near the entry are driven with a narrow face 3 m wide. To provide for the ventilation of the rooms they are connected by break-throughs at definite intervals. After the rooms have been mined out the rib pillars are robbed by driving cutting pockets (the pocket-and-stump method) and retreating towards the entry.

Experience gained in the USA shows that to ensure stability of the roof and uniform rock pressure the working faces of simultaneously

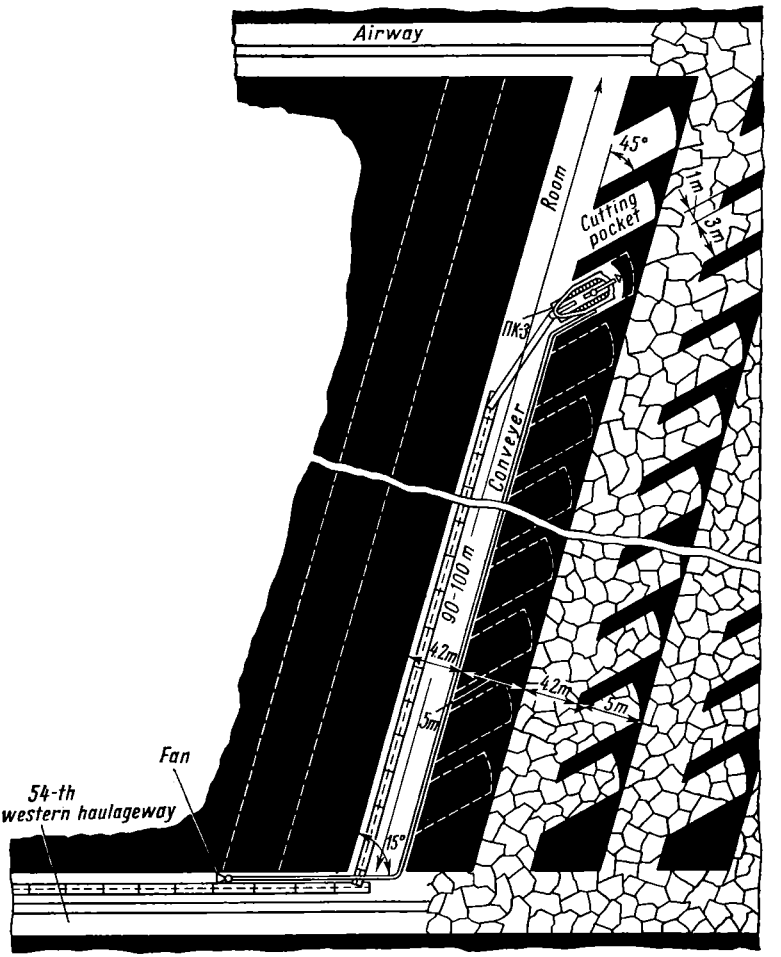


Fig. 106. Variant of room-and-pillar method of mining

mined rooms and, later on, those of rib pillars should follow a straight line running at an angle of 45° to the room axis.

The high labour efficiency and mining intensity attained by using this method were due to the wide-scale utilization of mechanical equipment with appreciable losses of coal.

Coal in the rooms is undercut by cutting machines designed to operate in short-run faces. These machines are mounted on self-propelling bogies and move easily from one face to another, taking care of up to six-eight faces with an aggregate breast area of up to 100 sq m during one shift. The undercut coal is broken off and loaded by loading machines onto a conveyer or directly into mine-cars. In coal mines of the USA extensive use is made of various types of self-propelling mine equipment.

An important drawback of the room-and-pillar method practised in American mines is the high loss of coal reaching 30-35 per cent. This loss occurs chiefly in pillaring.

The scope of room-and-pillar mining in the coal mines of the USSR is limited because the mining and geological conditions do not permit its application in the major coal basins and regions of this country. When the conditions are favourable (predominantly horizontal, or nearly horizontal occurrence of the bed, stable floor and roof, seams at least 1.5-2 m thick) the room-and-pillar system may be used provided effective methods for recovery of rib (intervening) pillars are introduced which will ensure the reduction of coal losses to the permissible level (15-20 per cent).

In the USSR the total proportion of coal mined by this method is very small. The room-and-pillar method was applied on a pilot scale at the Butovka-Donetskaya mine back in 1964, it showed better results as compared to the breast-stoping and pillar methods. The principal features of the system are seen in Fig. 106. The output per man per shift reached 18 tons (with ordinary systems it was 7 tons), the net mining cost per 1 ton was 1.07 roubles (with ordinary methods it was 1.85 roubles). Coal losses did not exceed 25 per cent. The minable thickness of the seam was 2.4 m. The prepared pillar was worked out from top downwards. Rooms alone were timbered (two props under butt caps), the spacing between the timber sets being 1 m. The cutting pockets were not supported, except for the room and the emergency exit insets, which were maintained by reinforced support and organ timbering. Abandonment of pillars between the cutting pockets allowed smooth subsidence of both the immediate and main roof. The organ timbering served as a break line for caving of the roof.

The technical potentialities of the room-and-pillar method are far from being exhausted.

4-10. CYCLIC STOPING AND METHODS USED BY INNOVATORS

Of prime importance in ensuring normal operation at the faces of productive and development workings is proper organization of work, and this can be achieved only by systematically adhering

to pre-set work cycles. A cycle in the production face is the aggregate of all mining operations performed within a certain period of time in an appropriate sequence to secure a definite rate of face advance. For instance, in mining by the coal-wall-level method a cycle consists in the drawing of coal throughout the length of the wall and over an advance distance stipulated by the technical schedule. In combine- and machine-serviced faces the advance rate per cycle is taken to be the depth of the cut, or the combine working range, while in working faces of steep seams it is the width of two support sets.

Cyclic organization of work is the simplest and most convenient when the sequence of operations is timed with work shifts and 24-hour periods. At present, some of the coal mines in the USSR operate on a tighter schedule of several cycles per 24 hours. Strict adherence to the cyclic operation schedule is the necessary pre-requisite for the fulfilment and overfulfilment of the coal-wall output plan.

Cyclic operation ensures optimal conditions of work for the miners and enhances their safety, since in this case precautionary measures are of a permanent nature, which is the best guarantee against accidents.

With cyclic organization the established order of work in the face may be presented in the form of a cyclic operation chart, which includes a so-called planned-work diagram and a labour distribution chart.

The planned-work diagram (Fig. 107) indicates the distribution of face work in time and space. The duration of the cycle in hours is plotted to a definite scale on the horizontal axis, and the length of the wall, on the vertical axis. Individual wall operations are marked out on the planned-work diagram by conventional lines or hatched areas with allowance made for the time and place of these operations. Rectangular areas denote operations to be performed simultaneously over the entire length or a certain portion of the face, whereas lines indicate operations conducted during each period of time at some point of the face.

The diagram clearly demonstrates the sequence and coordination of operations in time and space.

The second constituent part of the cyclic operation chart is the labour distribution chart. The cyclic operation chart also contains a table of technical and economic characteristics in conformity with the planned cyclic organization of mining operations. These characteristics are required for working out plans and estimates, and also for the assessment of the face performance.

Figure 107 represents a planned-work diagram for July-August 1967 in coal wall No. 33 of Karagailinsk mine No. 1-2 (the Kuznetsk coal basin) worked with the use of a mechanized set of equipment

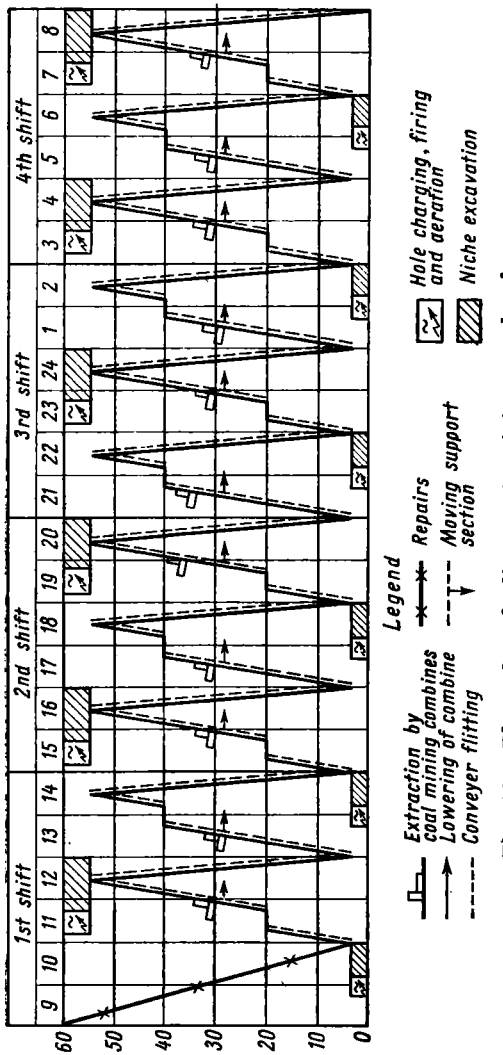

1st shift
2nd shift
3rd shift
4th shift
9 10 11 12 13 14 15 16 17 18 19 20 21 22 23 24 1 2 3 4 5 6 7 8
60 50 40 30 20 10 0
Legend
Extraction by coal mining combines
Lowering of combine
Conveyer flitting
Repairs
Moving support section
Hole charging, firing and aeration
Niche excavation

Fig. 107. Planned-work diagram in mining a coal wall -

(ОМКТ-м support, КШ-1 continuous coal-miner, and СП-53 conveyer) in which up to 11 cycles were performed each 24 hours. This planned-work diagram is significant, for it bears proof of great possibilities offered by integrated mechanization of stoping operations. In this particular case mining operations proceeded in 4 shifts in a coal wall 60 m long, the seam thickness being 2.9-3.0 m and its angle of pitch 12-15°. The coal was drawn from cut-outs by drilling and blasting. The amount of coal extracted per cycle was 150 tons. The advance rate of the coal wall per month was 214 m. Output per faceman per shift reached 51.2 tons.

In addition to planned cyclic operation charts actual-performance charts are also compiled. A comparison of planned- and actual-performance charts makes it possible to reveal shortcomings, detect their causes and record the final results of the work done. The advance rate of the face per cycle is determined from the depth of the cut or the working range of the combine. The planned monthly advance of the face is set at 10-15 per cent below the rated one.

Apart from the effective organization of stoping it is necessary to make provision for stand-by faces. They are particularly important in mining with coal walls of considerable length. An active coal wall may become inoperative as a result of incompetent conduct of mining operations or because of geological disturbances. The stand-by face footage should comprise 20-30 per cent of the total active walls. The stand-by faces must be so maintained that mining could be started immediately.

All the basic points concerning cyclic stoping, which have been set forth with special reference to the continuous method of mining, are also fully applicable to all other systems.

A further step forward in the organization of stoping is operation according to a cross-flow chart [1]). Elaboration of such a chart (for each working face) is preceded by a detailed description of all operations performed in the coal wall, from the beginning to the end of the cycle, along with time records. These data form a basis of the cross-flow chart. An example of a cross-flow chart is the chart used at the Novo-Pavlovskaya mine in the Donets coal basin (Fig. 108).

The legend for individual types of operations, their possible routing and the time required for their completion is given below. (К—end). 0-1—shift change, 1-2—prophylactic inspection of the mining combine, 2-3—cutting, 2-4-5—shifting of posts, 5-6—cutting the upper portion of the coal wall, 6-7—passage of coal from the cutout, 7-8—cutting the wall above the cutout, 8-9—shifting anchor props, 9-К—lowering the mining combine, 1-11-12—timbering

[1]) In the USA and England this bears the name of the PERT (program evolution review technique), time and labour chart.

the wall, 12-13—hauling timber from the entry to the coal wall, 13-14—timber delivery to the breast of the face, 14-K—lowering timber down the coal wall and its distribution, 9-10—replacing spontaneously knocked down pieces of timbering, 10-K—setting up safety platforms in the coal wall, 5-18—digging hitches for

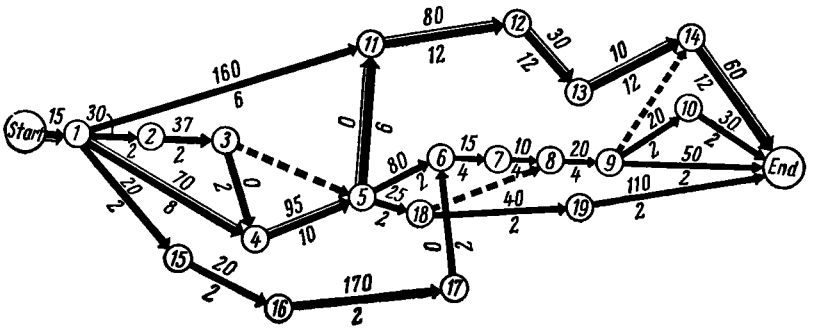


Fig. 108. Cross flow chart for a 200-m long wall (1 cycle per shift) in a steep seam worked by a YKP continuous coal combine at the Novo-Pavlovsk mine (Donets coal basin)

anchor posts, 18-19—transferring the lower crib, 19-K—shaping the lower cutout, 1-15—clearing organ timbering near the top pillar, 15-16—transferring the top crib, 16-17—shaping the top cutout.

Routes	Time, min	Time reserve, min
0—1—2—3—4—5—6—7—8—9—10—K	352	8
0—1—2—3—4—5—6—7—8—9—K	352	8
0—1—2—3—4—5—18—19—K	352	8
0—1—4—5—6—7—8—9—10—K	355	5
0—1—4—5—6—7—8—9—K	355	5
0—1—4—5—18—19—K	355	5
0—1—4—5—11—12—13—14—K (critical route)	360	
0—1—2—3—4—5—11—12—13—14—K	357	3
0—1—15—16—17—6—7—8—9—10—K	320	40
0—1—15—16—17—6—7—8—9—K	320	40
0—1—11—12—13—14—K	355	5

Circles on the chart designate "events"—the intermediate or final results of the operation performed; the arrows indicate technological operations. The figures above the arrows: the denominator signifies the time in minutes necessary for the performance of the

work, the numerator—the number of men engaged in this operation. The dotted line indicates transfer of the labour force from one operation to another. The solid line shows the actual work done.

The critical route on the cross flow chart covers the most time-consuming operations. On the drawing, the critical route (designated by a double line) includes the events 1-4-5-11-12-13-14-K (termination of the work). It is equivalent to 360 minutes, e.g. to one work-shift.

In drawing up the chart consideration is given to the travel speed of the mining combine along the face, the maximal load on the other equipment, etc.

The cross-flow chart is first discussed by the mine foremen, shift and gang bosses, and other members of the operating and supervisory staff, who make suggestions and corrections, and only then is it considered at the plenary meeting of the technical and economic council of the mine.

After the chart has been approved the miners are familiarized with it.

At the Novo-Pavlovskaya mine the cross-flow chart serves as a basis for the rhythmic and economic organization of work. It helps to make rational use of the available reserves of equipment, materials and manpower.

Operations along the critical route are personally followed up and supervised by the mine section chief. At the Novo-Pavlovskaya mine the critical route embraces timbering of the coal wall, the supply and lowering of mine-timber.

The fulfilment of the cross-flow charts in working faces is checked every day during the "dispatcher hour", usually at the beginning of the first shift. The mine dispatcher informs the duty foreman and the section chiefs of all failures and disruptions in the "events". In exceptional cases the dispatcher reports to the mine management.

The advantages of the cross-flow charts cannot be overestimated. Their main merit is pictorialness. As distinct from the usual linear charts they demonstrate the complex interrelationship of all operations and indicate the sequence and concurrence of their performance.

Cross-flow charts should not be regarded as hard-and-fast standards. They are continually perfected with changes in mining conditions and improvements in the organization of labour.

4-11. METHODS OF MINING THICK SEAMS

General

Many coalfields of the USSR have thick beds (more than 3.5 m). Drawing coal from them is very complex. Even now coal losses in working thick beds are relatively high, reaching 20 per cent and

more. The difficulty of reducing these losses lies in combining full recovery with a reasonable net cost of mining operations.

The choice of a mining method is also complicated by the presence of serious fire hazard, which is especially to be reckoned with in working thick coal beds.

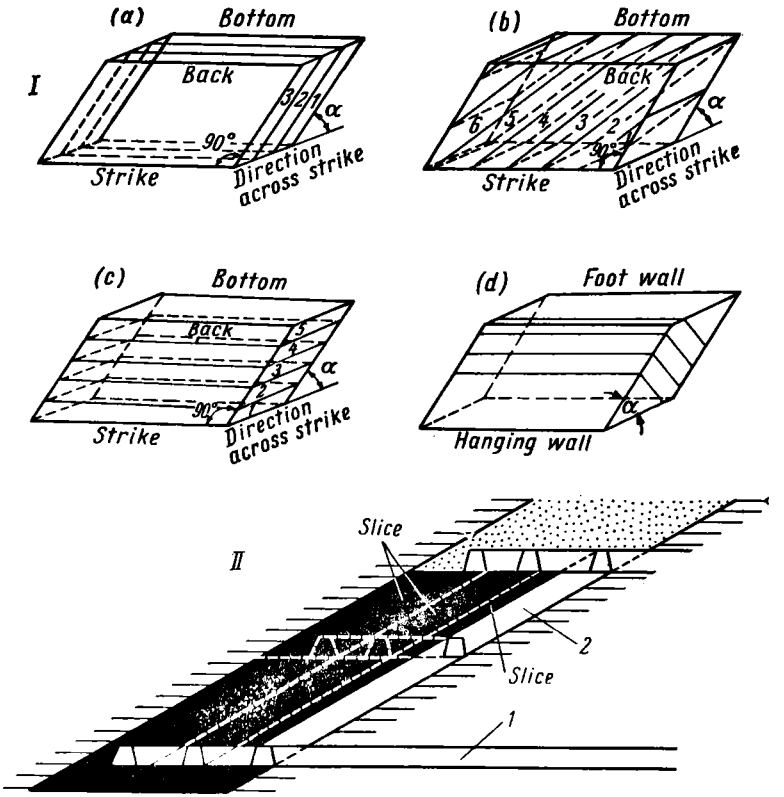


Fig. 109. Slicing methods:

I—position of slices: *a*—inclined slices; *b*—diagonal slices; *c*—horizontal slices; *d*—transverse slices; *II*—division of a thick seam into inclined slices: *1*—crosscut; *2*—gravity incline or slope

The methods of mining thick seams are classified into two main groups: with and without division into slices (Fig. 109). The principle of coal winning from thick beds by slicing consists in that a seam is worked out gradually in slices whose thickness is so chosen that it is convenient to work in them. The adopted slice thickness is 2 to 3.5 m.

Slicing Methods in Coal Beds

Inclined Slicing. Inclined slices may either represent individual benches of a seam, if they have a regular structure and suitable thickness and are split by interlayers of gangue (measuring 0.5 m and more) or comprise a bed of uniform structure artificially divided into several inclined slices (Fig. 109).

Development openings (entries, gravity inclines, slopes) are so situated as to enable each slice to be mined as a separate seam of medium thickness. The slices may be worked out in ascending order (from the bottom to the back of the bed) or in descending order (from back to bottom). In the former case filling is mandatory, whereas in the latter either stowing or induced caving of rocks may be used, depending upon the actual mining conditions. The presence of plastic clay shales in the roof favours the descending order of slicing-and-caving. The presence of strong rocks complicates extraction with roof caving. The drawing of the slice underneath caved-in rocks should not be started until after a certain period of time required for the rocks to consolidate. The compaction of rocks slows down when coal pillars are abandoned during extraction of the slice. In this case the abandoned solid mass of coal also exerts a concentrated pressure on the timbering of the subjacent slice. Gangue intercalations facilitate the extraction of individual slices in a thick bed. In the absence of such intercalations and in downward successive slicing the bottom of the slice being mined is covered with preliminary support pieces underpinned by the timbering of the underlying slice. More detailed information on the laying of flooring is given in the description of horizontal slicing of coal seams.

Inclined slicing-and-caving is customary with a moderate dip (10-35°) in seams not exceeding 8-10 m in thickness (not more than four slices) with a relatively regular structure and mode of occurrence. In the presence of barren rock intercalations with somewhat larger angles of pitch are admissible.

In inclined slicing-and-filling the method under discussion may be applied in mining beds with an angle of dip in excess of 30° (approximately up to 60°). In this case the ascending order of slicing can be used only in beds up to 8-9 m in thickness (three slices). The last slice may be extracted with caving. The ascending order of slicing cannot be used in the following conditions: in seams with soft and friable coal susceptible to spontaneous caving during longwall mining, in sections with disturbances or extremely jointy coal.

If the descending order of slicing with tight filling is used the thickness of the seam may be quite considerable. When choosing a mining method for a high steep seam one should compare systems involving inclined and horizontal slicing. The method of horizontal

slicing is described below. With inclined slicing each slice is worked out as a medium-thick bed with long pillars on strike.

Owing to the substantial advantages of mechanized sets of equipment thick gently sloping seams can be mined in long pillars with

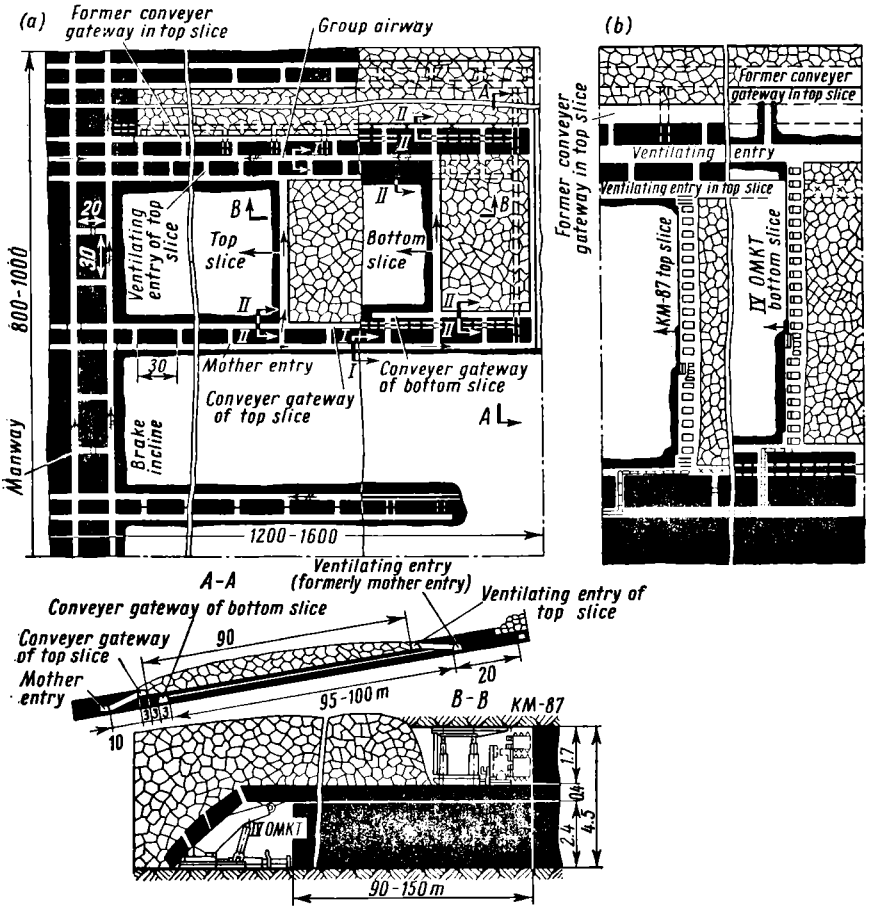


Fig. 110. Inclined slicing with coal drawn in slices by mechanized sets of equipment:

a—general view of system; *b*—schematic representation of walls in top and bottom slices

the bed divided into two slices (if it is up to 5 m thick), each slice being worked by a mechanized set. Such a system is shown in Fig. 110. Investigations carried out at the Skochinsky Research Mining Institute proved it possible and expedient to work out a

gently dipping bed 3.5-5 m thick by a single mechanized set with hydraulically-operated support (such a mechanized set can be designed and constructed). In this case the use of long pillars on strike with panelling of the mine-field is expedient (Fig. 111). Before such a set is built the mining of a thick (up to 10-15 m) gently dipping bed

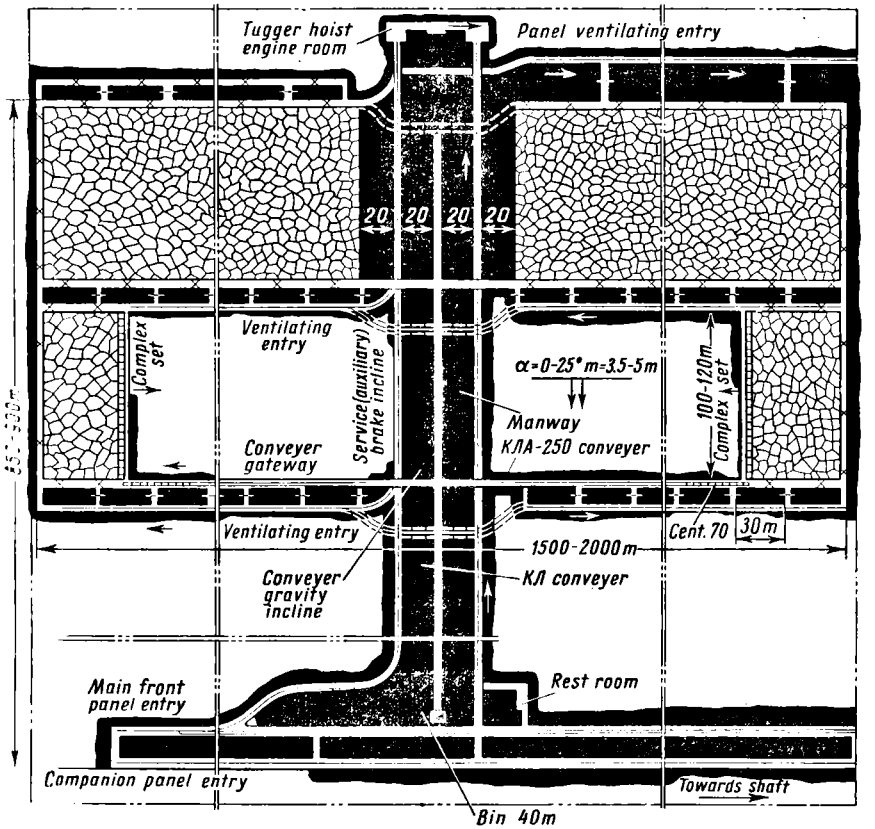


Fig. 111. Long-pillar method of mining a bed 3.5-5 m thick by using full-seam mechanized set of equipment

may be done in two slices with a mineral mass left in-between, by employing a special КТУ-2м set of equipment. This set is intended for working slightly sloping (up to 15°) beds 6-15 m thick. Extraction is done in two slices, the interslice mass being drawn through special openings in the mobile support of the lower slice. Roof control is effected by complete caving. The КТУ-2м set includes flexible cooling, hydraulically operated support of the fencing-off type, an

СП-63 conveyer, and a reloader. The mobile support is shifted by means of hydraulic jacks along the floor of the bed under the protection of a flexible metal roofing laid down on the bottom of the upper slice mined by long pillars on strike. The flexible roofing consists of flat strips 50 mm wide and 3 mm thick, and of wire netting.

Commercial tests and wide introduction of the КТУ-2 sets began at the Tomusinsk coal mine in 1961 and are still continued. Over the period from 1961 to 1965 a total of 3.9 million tons of coal had been produced with these sets. In these mines thick slightly dipping seams are extracted. They are worked by ordinary inclined slices

	Inclined slicing with the use of КТУ-2 set	Inclined slicing with individual support pieces
1. Average monthly yield of coal wall, tons	10,926	4,314
2. Labour productivity per man per shift in producing section, tons	6	3.9
3. Length of development workings per 1,000 tons of coal output, m	7.7	11.0
4. Timber consumption per 1,000 tons of coal output, cu m	18.7	58.1
5. Prime cost of mining 1 ton of coal, roubles	3.86	5.14
6. Overall losses of coal, %	15.2	21.2

with individual support and with the use of КТУ-2 sets. The simpler technology of coal winning with the aid of КТУ-2 sets helped to achieve better technical and economic results, as may be seen from the comparison above.

Annual saving in expenditures per set equalled 191 thousand roubles.

Inclined slicing is frequently applied in working thick coal beds in the USSR. Where this system is inapplicable (heavy pitch, nonuniform or considerable thickness of the bed) horizontal slicing is used.

Horizontal slicing (Fig. 112). This method is employed in a variety of conditions, such as mining thick beds and deposits of different shape and thickness containing firm, soft, self-igniting and firedamp coal. The wall rocks may have any degree of stability. The seam may dip at different angles, although mining of inclined and steep beds is more convenient and effective. In horizontal slicing the level interval height is set at 40-100 m (commonly 60 m), the

height of the slice being 2.8-3.5 m. Large level interval heights are divided into individual sublevels. The slices in the block may be mined in ascending or descending order. The former is admissible only with filling of the goaf, while in descending order mining is done both with stowing and induced caving of the roof.

Splitting or cracking of coal presents great fire hazards.

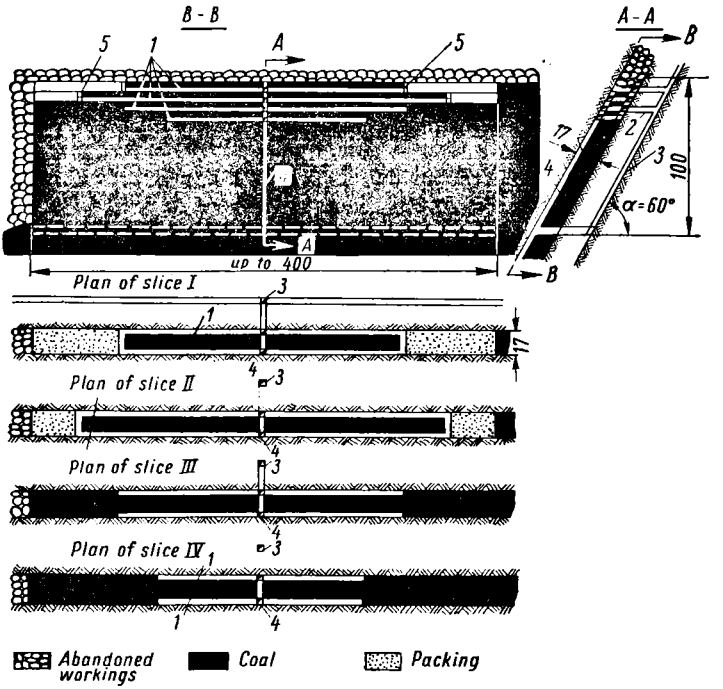


Fig. 112. Variant of descending horizontal slicing-and-filling:
1—slice entry; *2*—slice crosscut; *3*—gob-loading dump chute; *4*—coal chute; *5*—cut

When the level is worked from top down these hazards are markedly reduced. Hence the widespread use of downward successive slicing in mining coal deposits in the USSR and a number of foreign countries.

Particular emphasis should be placed on the expediency of this method in mining thick and steep beds, when the use of inclined upward slicing involves substantial difficulties, and also when mining high seams in curves of folds and recovering coal pillars left after working thick steeply pitching beds.

Upward horizontal slicing has turned out to be ineffective and unsafe, and for this reason is used hardly at all.

Below is given a brief description of the methods of descending horizontal slicing-and-filling.

A horizon or individual sublevels are divided into extraction sections (blocks) 200-400 m long. In each horizontal slice stoping may be done on strike or across the strike, depending upon the thickness of the bed, the equipment employed and the firmness of the roof. In the coal mines of the USSR the most widespread method is working out a slice with a continuous face pushed forward across the strike. The number of concurrently mined slices in a section (block) may vary from one to three, depending upon the target output and the mining conditions. With work proceeding simultaneously in several slices stoping in each overlying slice should be done somewhat in advance (by at least 20 m) of the subjacent one. This method requires a relatively large amount of preliminary and subsidiary development work. Figure 112 illustrates the method, indicating development and productive workings in winning coal through individual cuts made across the strike.

With the system under consideration the development openings include: a gob-loading dumping chute (upraise) in the middle of the extraction block, raised in the foot wall rock; a coal chute in the centre of the block, run in the contact rocks of the hanging wall; short slice crosscuts driven from the gob-loading dumping chute to the seam (one crosscut to serve several concurrently mined slices); slice entries near the hanging and foot walls of the seam (if the latter is less than 8 m thick only one entry is driven in the middle of the slice along its width); crossdrifts run in slices to ensure ventilation of slice entry headings.

The scope of development work with this method may be cut down by transferring the gob-loading dumping chute to the foot wall of the seam (elimination of slice crossdrifts).

In extracting coal by splitting (from cuts) drilling and blasting are employed, the coal being transported by conveyers.

Stope support. In horizontal slices preliminary and face timbering is used (Fig. 113). Preliminary timbering consists of sills and a board flooring. Sills, 12-20 cm in diameter, are placed normally to the stope in recesses made in the bottom of the slice being mined, with their ends (viewed in plan) laid side by side. Sills are usually spaced at a distance of 1-1.3 m. Sills carry a flooring made of boards 3-5 cm thick laid parallel to the breast of the stope. The face timbering is made up of timber posts 15-25 cm in diameter wedged under the cap sills of the preliminary support, or of wooden props and relief posts placed under the sills of the preliminary timbering. The rows of posts are spaced at 1-1.3 m.

To raise the effectiveness of stoping in this method the following improvements may be introduced: the use of thin iron strips or steel

ropes (instead of timber sills), of wire netting (in lieu of a wooden flooring) and of walking metal props (in place of wooden ones) with special arrangements for their withdrawal from the mine-fill. Breaking and loading of coal onto the conveyer can be mechanized by using light shuttle-type mining combines. To mechanize filling operations air-blast stowing machines can be used. Mobile powered

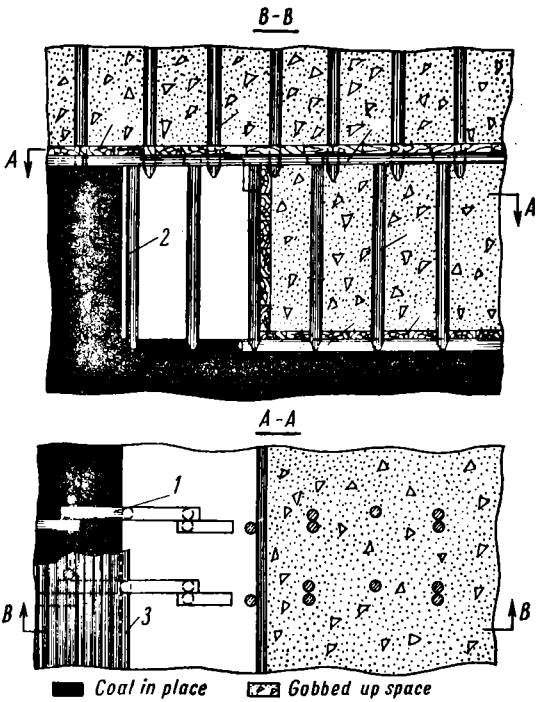


Fig. 113. Preliminary and face support employed in mining a seam by descending horizontal slicing-and-filling:
1—sill; *2*—post; *3*—plank

support can be employed in place of ordinary timbering. This support is shifted hydraulically in the wake of the advancing stope. Successful application of mobile support will greatly increase the effectiveness of the system.

As contrasted to inclined slicing the method of extraction by horizontal slices is more flexible and applicable in varied mining and geological conditions.

Methods of diagonal and transversely-inclined slicing. Like in horizontal slicing extraction of coal in a level with these mining methods is done by individual blocks. Within the block coal is drawn

in slices from bottom up in diagonal slicing and both from top down and from bottom up in transversely-inclined slicing. Downward mining of slices with float stowing is more efficient. Extraction of slices from top down is attended by caving of the roof and power stowing (Figs. 114, 115).

The adoption of diagonal and transversely-inclined disposition of slices was prompted by the desire to ensure gravitation of the

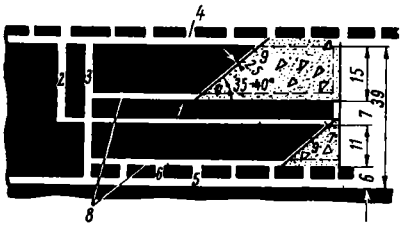


Fig. 114. Diagonal slicing (in the plane of the seam):
1—subentry sill pillar; *2*—gob-loading dump chute to the lower sublevel; *3*—coal chute; *4*—top entry; *5*—bottom haulage entry; *6*—rise entry; *7*—subentry; *8*—parallel entries; *9*—slices

coal and mine-fill in the production face (the slices being arranged at the corresponding pitch angle). However, despite the possibility of utilizing the weight of the coal and mine-fill for their transfer within the stope area these methods have a number of substantial

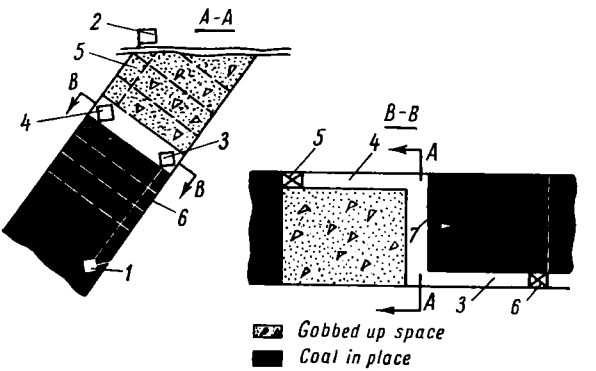


Fig. 115. Transversely inclined slicing:
1—haulageway; *2*—ventilating entry; *3*—slice entry for transfer of coal; *4*—slice-stowing entry; *5*—gob-loading dump chute; *6*—coal chute; *7*—production face

shortcomings, viz: a short stoping front (less than in horizontal slicing and considerably less than in inclined slicing); the difficulty of setting up heavy timbering and of using efficient equipment for breaking the coal; the inconvenience of operating this equipment.

Owing to these disadvantages, diagonal and transversely-inclined slicing is employed rarely and the future prospects for its use are very limited indeed.

Nonslicing Methods of Mining

Pillar Methods. *Long-pillar mining on strike in slightly inclined beds.* This method is practised in working out seams 3.5-5 m thick. The general pattern of preparing a high seam for stoping in the pillar methods is analogous to that used with medium-thick seams in long-pillar mining on strike. Each coal pillar, 7-9 m wide, is recovered by cuts or pockets made up the raise, timbered with high props and with coal stumps abandoned near their boundary. A variant of the system involves extraction of a pillar in strips 17-19 m wide down the dip with coal pillars 1.5-2 m thick left between the strips.

Coal from cuts is drawn with subsequent caving of the roof. This mode of stoping is called Silesian. Formerly it was widely used in the Silesian and Dombrowsky coal basins in working slightly dipping and inclined beds.

The method has serious drawbacks, such as extremely high coal losses (30-40%), and inconvenience of operation: high working faces; heavy, long timber props; rolling down of big coal lumps.

Labour expenditure in timbering can be reduced by using suspension roof support (rock bolting).

The coal losses due to roof control by caving decrease when hydraulic fill is used.

Long-pillar mining to the rise with hydraulic fill. This method has become very popular in Polish coal mines (Fig. 116). A panel 300-400 m long is prepared by running three strike entries on the haulage level and two entries on the airway horizon, along with a number of gravity inclines spaced at 60-75 m.

Some of the entries are designed for handling hydraulic fill and water runoff. In the production face and along the gravity inclines the coal is delivered by conveyers. The stope area is supported by timber sets running parallel to the face breast and spaced at 1.2 m. The headboards are 4-6 m long. Props under the headboards are set up every 1.5-1.2 m. The space interval for filling is 8-10 m.

Long-pillar mining on strike in a steeply pitching bed. These methods are commonly employed with caving of the roof and less frequently with stowing of the worked-out space. In mining with caving of the roof coal losses are high and building up support in the top portion of the working face with the use of heavy roof timber pieces presents quite a problem.

Because of their serious shortcomings the methods of mining by long pillars on strike have not found wide application in the USSR.

Methods of mining by long pillars to the rise extracted down the dip with shield support designed by N. Chinakal.

From the foregoing it follows that many systems used in mining high steep seams are far from perfect. In endeavouring to improve the

methods of mining thick steeply pitching beds Soviet specialists designed movable metal support in the shape of shields.

The principle of movable shield support is well known: once the working place has been fixed within the bounds of a pillar the production face is pushed forward together with the installed support without any further expenditure of time and money for setting up

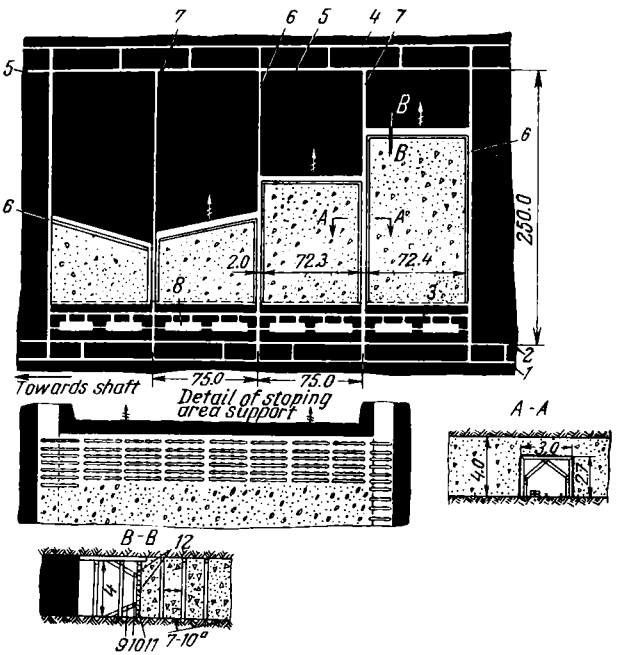


Fig. 116. Mining a thick gently sloping bed by long pillars up raise with hydraulic fill:

1—haulageway; *2*—offtake entry; *3*—water-collecting entry; *4*—ventilating entry; *5*—pulp supply entry; *6*—air raises; *7*—pulp supply gravity inclines; *8*—settling basins for water clarification; *9*—knee brace; *10*—girder; *11*—boards; *12*—timber slabs

a new one; the weight of the support and the rock pressure are used for shifting the support. The shield method as proposed by N. Chinakal is described below.

The essence of the method and the shield design. With this method (Fig. 117) the hard, hazardous and unproductive labour of the timberman gives way to easy and safe shield control. The mining practice in the Kuznetsk coal basin shows that shield support greatly increases labour productivity, cuts down consumption of mine timber and makes for safe working conditions. The following shield designs are used: light single sectionalized shields, heavy single sectionalized

shields, twin sectionalized shields; flexible non-sectionalized shields. Let us now consider systems and shields of different design.

The system with a single sectionalized shield, first proposed in 1938 by N. Chinakal, essentially consists in the following. With the shield mining method (Fig. 117) a level is divided into panels 200-250 m long. On the main horizons a panel is opened up through crosscuts, a haulage entry being driven from them. The ventilating

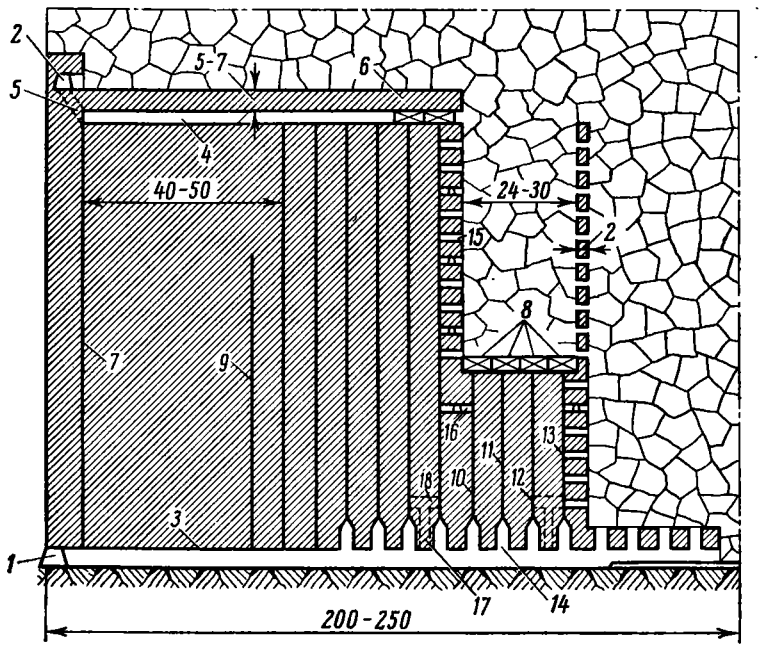


Fig. 117. Method of mining by long pillars to the rise with stoping down the dip and shield flooring (shield mining):

1, 2—intermediate crosscut; *3*—haulage entry; *4*—ventilating entry; *5*—inclined through-cut; *6*—level pillar; *7, 9*—boreholes; *8*—4-section shield; *10, 11, 12, 13*—coal chutes; *14*—pocket; *15*—service raise; *16*—breakthrough; *17*—ventilating through-cut; *18*—breakthrough

entry is usually run from an inclined through-cut below the elevation mark of major level workings of the overlying horizon, and a level pillar of 5-7 m is abandoned. The haulage- and airways are connected by boreholes, whose section is enlarged to conform with that of the through-cuts. The panel is mined by pillars 25-30 m or more in width. These are recovered starting from the panel boundaries towards the crosscuts. On the side of the drawn pillar a solid coal stratum about 2 m thick is left. The stope in the pillar lies under the shield. To prepare the pillar for extraction coal dump chutes

are run in it every 6 m by reaming the earlier drilled boreholes 390 mm in diameter to 850-900 mm (Fig. 118). In their lower part coal dump chutes are enlarged to 1.5 × 1.5 m, forming a pocket. To secure access of workers to the stope area a ladderway is provided in the adjacent pillar under preparation, which is connected by breakthroughs with the coal dump chute every 5-7 m. In the coal dump

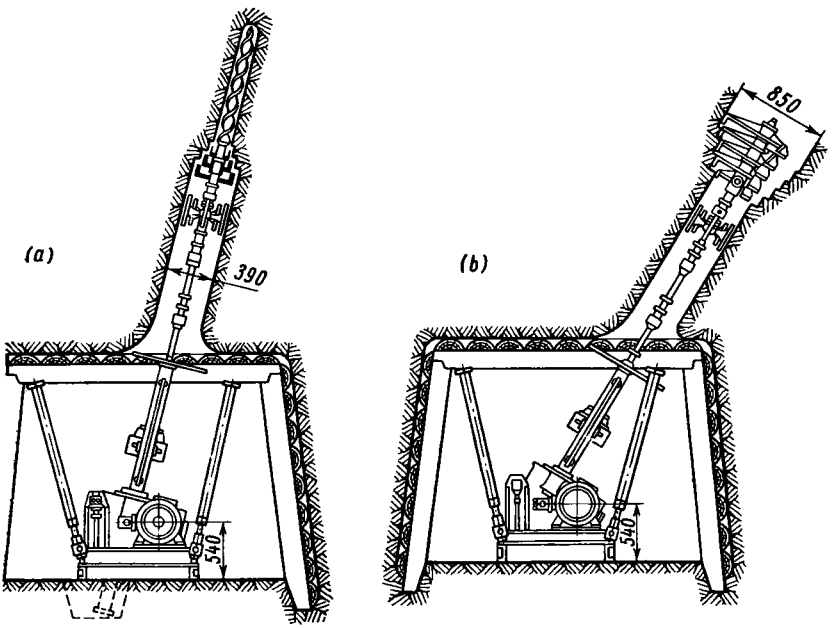


Fig. 118. Diagram illustrating operation of the СБМ-3У machine:
a—drilling; *b*—reaming

chute a ladder is suspended and another ladder is mounted in the last dump chute of the pillar under extraction. These ladders are designed for communication with manways and breakthroughs. The ladders are suspended from the shield and descend together with it. The manways are timbered with three-piece sets and from them exploratory crossdrifts are run to define more exactly the thickness and pitch of the bed. Since the pockets are filled with coal the aeration of the stope is realized via specially driven workings above the pockets (on the diagram they are shown by dotted lines). When the preparation of the pillar for stoping is completed a chamber for the assembly of the shield is excavated. A flat sectionalized shield up to 6 m long, whose width is 0.5 m less than the thickness of the bed, comprises a steel frame made of No. 20-30 channel bars and angles.

The frame is covered with a dead flooring composed of several rows of rough logs (their number depends upon the pressure the shield is expected to bear). The dead flooring is bolted to the steel frame. The frames of individual sections are tied up by means of a four-strand steel rope 25 mm in diameter (flexible connection). 15-20-cm clearances between individual sections are overlapped by a steel

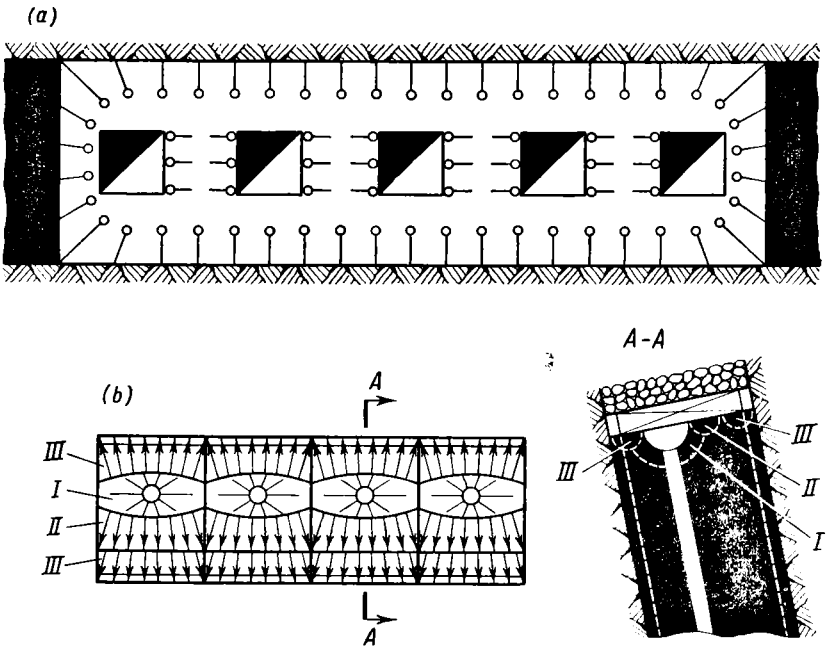


Fig. 119. Rounds of holes and sequence of their firing

netting laid down in two layers. In addition to the shield section a protective grating is suspended, which overlays the coal dump chute and moves together with the shield. As stated above, metal ladders are suspended in the marginal through-cuts in addition to the protective gratings. Upon completing the assembly of all shield sections of the production pillar the first shield sections of the next pillar are mounted. Prior to stoping a protective rock cushion is built up on the shield by blasting the floor pillar and then transferring the rock from an earlier mined-out level. As the shield is moved on the pressure upon it declines owing to friction forces, which diminish the weight of the rock column overlying the shield. Coal under the shield is broken by blasting holes 1.2-2 m deep. The pattern of the holes and sequence of their firing are shown in Fig. 119. During

operation the shield should be placed normally to the back and bottom of the working face; to do this, at the start of shield operation the coal is drawn so as to allow greater penetration of the shield into the bottom of the bed. The stoping is completed at a distance of 3-4 m from the haulage entry and the shield is not dismantled as

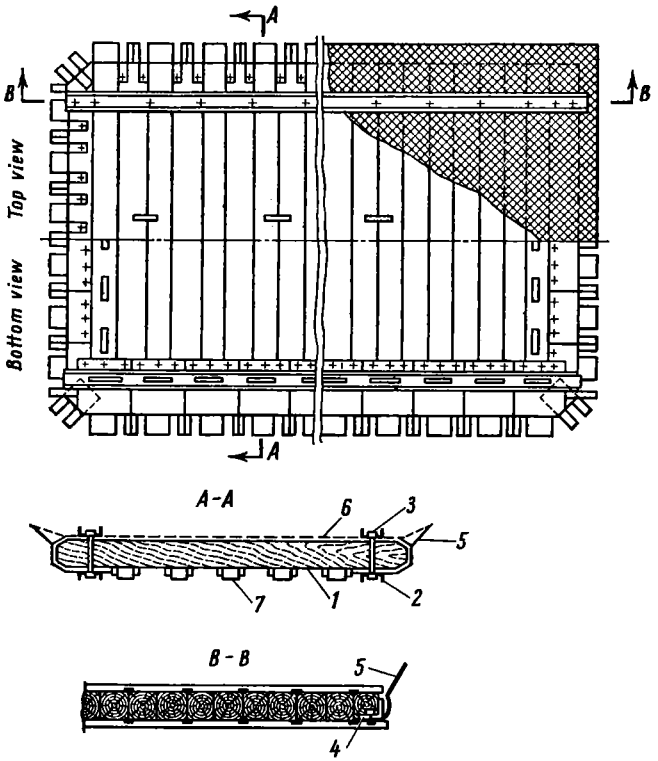


Fig. 120. Design of a flexible integral shield

a rule. In beds whose thickness exceeds 8 m shields with a reinforced frame (heavy sectionalized shields) are employed. In seams 9-12 m thick twin sectionalized shields are used. Beds 11-16 m thick are worked in slices, using 2 rows of single sectionalized shields and leaving small coal pillars 2 m thick between the rows. To cut down timber and metal consumption flexible non-sectionalized shields (Fig. 120) are widely used. The flexible non-sectionalized shield, up to 60 m on strike, comprises one row (*1*) of dead flooring placed across the strike. The rough logs of the flooring are tied by girths consisting of two pairs of channel bars (*2*) running on strike. The channel bar

girths are tied by cover plates and bolts. They are held together by bolts (*3*) extending between the squared rough logs of the flooring. At the butt ends of the shield roofing the lower girths of the channel bars are linked together with cross beams (*4*) made of No. 20 angles. To protect the butt ends of the flooring from any damage during the movement of the shield, aprons (*5*) made of sheet steel 3 mm thick are fastened along the entire perimeter of the shield roofing. Anchors are fixed to the round lagging of the floor to which wire cables of protective gratings, ladders and control ropes are fastened. The dead flooring is covered with double-row metal netting (*6*). Metal consumption per 1 sq m of a shield of sectionalized design for a bed 5.5-7.5 m thick amounts to 100-110 kg; if the shield is non-sectionalized the consumption reduces to 45-50 kg. Timber consumption is cut down 3-3.5 times and comprises 0.35-0.55 cu m per 1 sq m of the shield for non-sectionalized support. Flexible non-sectionalized shields of the described design are applied in steep seams 2-2.5 to 5-6 m thick. In beds from 5-6 to 7-8 m thick use is made of heavier shields made up of twin beams laid across the strike.

Practical application and proposed improvements of the shield mining method in the USSR. The shield method of mining has found the widest application in the Prokopyevsk and Kiselevsk districts of the Kuznetsk coal basin, where its proportional use accounts for up to 70% of the entire coal output, reaching 80-100% at individual mines. In the near future this system will be employed not only in the Kuznetsk basin, but in other regions as well. It is believed that shield mining will find application not only in seams up to 10 m thick and with an angle of dip over 55°, but also in thicker beds dipping at smaller angles.

N. Chinakal has proposed a new design of a shield on rollers to be used for an angle of dip less than 55°. Such shields should be employed in beds 4-8 m thick with firm wall rocks and hard coal. Of late, heavy shield support moved on by hydraulic jacks with the coal hauled by scrapers to the dumping chutes has been manufactured in the Kuznetsk coal basin. To reduce coal losses, which amount to as much as 20-25% and in individual instances even more with the system used at present, N. Chinakal has proposed that at first, before starting to draw coal from the production pillar, coal stumps between production pillars be robbed and the mined-out space be filled with rock cemented by a cheap binding material. The use of such mounts for the shields will greatly reduce coal losses. At present, work is underway in the Kuznetsk basin to improve the shield method of mining with a view to the introduction of filling by flushing; the stow moves and is added as the shield descends. This should appreciably reduce fire hazards and enable the system to be used in mining deep horizons.

In recent years shields adapted for mining steep seams of limited thickness (within the range of 1.5-4 m) have been subjected to field trials and introduced at coal mines in the USSR.

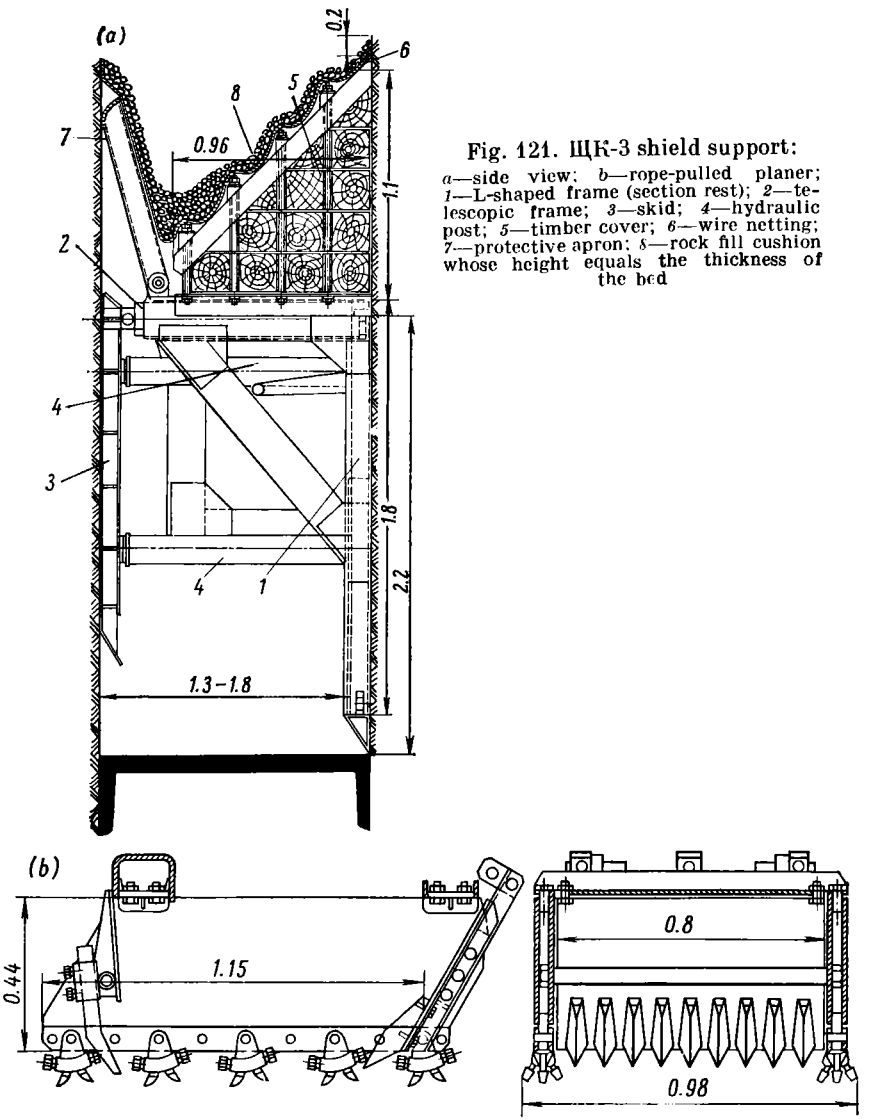


Fig. 121. ЩК-3 shield support:
a—side view; *b*—rope-pulled planer; *1*—L-shaped frame (section rest); *2*—telescopic frame; *3*—skid; *4*—hydraulic post; *5*—timber cover; *6*—wire netting; *7*—protective apron; *8*—rock fill cushion whose height equals the thickness of the bed

The ЩК-3 shield support designed by E. Makhno is adapted for mining long pillars or strips down the dip in low and medium-

high steep seams. The ЩК-3 shield set (Fig. 121) consists of hydraulically powered shield support and a rope-pulled plough, placed beneath the shield. The coal is extracted and hauled to the board by the plough pulled along the working face by a tugger hoist. Besides the shields of the above-described designs, an arch shield support designed by engineer G. Puzyrev has been tested.

It is expected that the shield principle will soon be widely used in mining not only coal, but also other mineral deposits.

Room methods of mining. The room mining method can be applied with strong enclosing rocks and various angles of dip. It is commonly used in beds with a flat pitch, where rooms are made up to 100-150 m long and 10-15 m wide with permanent intervening pillars 6-12 m thick left between them. In individual instances rooms are made 6 m wide and the intervening pillars 3 m wide. Rooms are extracted either by the advance or the retreat method with full or partial shrinkage of the mineral. In retreat mining the coal is drawn after a special working (crosscut) for the safe passage of men and supply of material and equipment to the working face has been driven along the axis of the room.

In working steep seams ranging from 5-6 to 10-12 m in thickness the room method is frequently used with suspension roof support of the hanging wall. In this version a slice of coal 2-3 m thick is first drawn near the back of the seam along the contours of the prospective room.

As this slice is worked out the wooden face timbering is set up and, at the same time, the back of the seam is held in place by installing suspension roof support. Once the top slice has been extracted the remaining portion of the room is mined with shrinkage of the coal (Fig. 122). The rooms are excavated 10-15 m wide, the intervening pillars being 3 m wide. The coal in the room is broken down by firing blast-holes. From the rooms the broken coal is delivered to the haulage entry by gravity, through funnels and chutes.

High losses of the mineral, which reach 40-60%, are a major shortcoming of room mining. Because of extremely high losses and fire hazards in underground workings room methods are rarely used in the coal mines of the USSR. In the Polish People's Republic room mining is attended by hydraulic stowing of rooms which brings the overall losses of coal down to 20%.

In the USSR room methods of mining find wide application in working deposits of rock salt and building materials. In mining rock salt deposits it is necessary to abandon permanent pillars, since water should not be allowed to penetrate into the stope area from the superposed rock strata. Such reliable protective pillars also rule out any possibility of caving-in or subsidence of superincumbent rocks. As an exception, room mining is also practised in working

other mineral occurrences whenever it is necessary to safeguard the integrity of the ground surface and of the overlying rock strata.

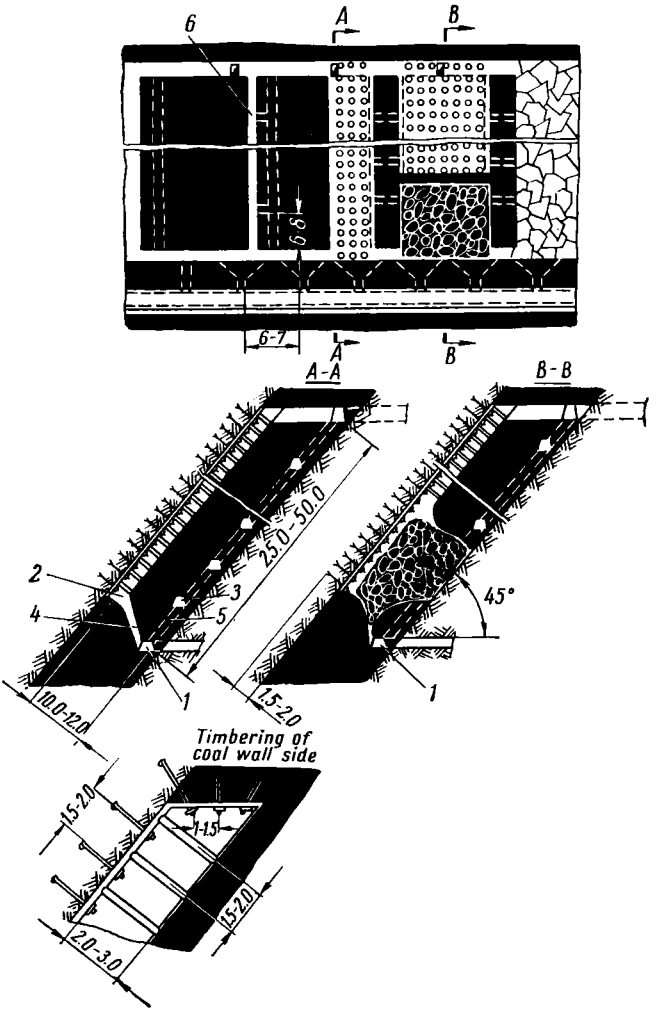


Fig. 122. Room mining of thick steep beds with top benches of the seam extracted by walls on the strike with the use of suspension roof support (section across the strike):

1—haulage entry; *2-3*—service entries; *4,5*—rise entries; *6*—cut-out raise

Basic information on room mining with special reference to working rock salt deposits is given below.

Rock salt occurrences, usually horizontal or slightly dipping, are mined by rooms on strike or across the strike with the shafts located centrally. The rooms are aerated through breakthroughs connecting them, with the air current moving in the general direction from the hoisting to the ventilating shaft (Fig. 123).

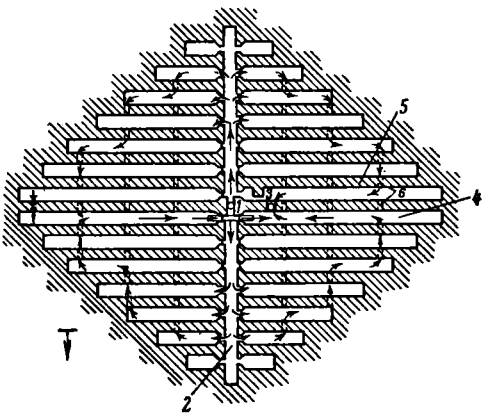


Fig. 123. Dispositon of rooms in mining a rock salt deposit (plan view):
1—hoisting shaft; *2*—main entry; *3*—air shaft; *4*—room; *5*—intervening pillar; *6*— ladderway; arrows indicate direction of air current

In selecting room mining methods it is important to determine the width of intermediate pillars, which can be estimated by using L. Shevyakov's formula:

$$SHq + shq_1 \leqslant \frac{sR}{n}$$

where S = horizontal cross-sectional area of ground per support pillar;
H = distance from support pillar top to ground surface (Fig. 124);
h = height of support pillar;
s = horizontal cross-sectional area of support pillar;
q = average unit weight of overlying rocks;
q_1 = unit weight of support pillar rock;
R = compressive strength of support pillar rock;
n = margin of safety adopted in calculating dimensions of support pillar.

The compressive strength of support pillar rock (kg/cm^2) may be set at 300-450 for rock salt, 270-360 for sylvinite, 60-160 for carnallite, 980 for sandstone, and 600 for dolomite.

In order to estimate the value of R more precisely a special compression test is needed.

In calculations the margin of safety n may be put down at 2.5-3.0, taking a static load as a basis.

The calculated width of the pillar should be increased by 25-30% to take account of the roughness of the side-walls and possible appearance of fissures and cracks on blasting. [1]

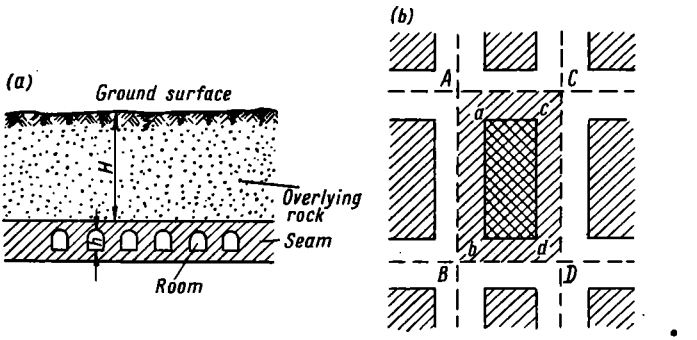


Fig. 124. Diagram illustrating estimation of the dimensions of a support pillar: *a*—vertical section of deposit; *b*—disposition of support pillars in plan; *abcd*—support pillar area; *ABCD*—overlying rock area per support pillar

By way of example let us consider the room-mining system used at one of the rock-salt occurrences of the USSR (Fig. 125). The rock-salt bed lies at a depth of 150 m from the ground surface, it is 30-40 m thick and has an average angle of dip of 4°.

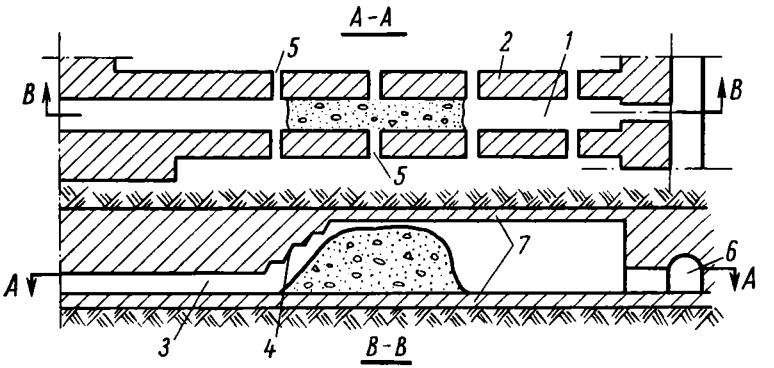


Fig. 125. Winning rock salt in a room:

1—room; *2*—intervening pillar; *3*—undercut working; *4*—"overhand" stope; *5*—manway-breakthrough connecting neighbouring rooms; *6*—main drift; *7*—rock salt pillar (cap and floor solid salt in place)

Arched rooms are arranged along the strike, their length being 100 m or more, span 17 m, and height 25-27 m.

[1] For more details on the estimation of intervening pillars see L. Shevyakov's book *Fundamentals of Projecting Coal Mines.* Ugletekhizdat, 1958.

In the roof and floor of the rooms a skin of salt 1-3 m thick is left. The dimensions of the pillar between through-cuts are 8 × 30 m. The through-cuts between the rooms are driven every 30 m. The salt is broken down in overhead stopes with provisional shrinkage of salt in the room. The salt in the shrinkage area serves as a stage for drillers.

The blast-holes in the room are drilled with electric augers, a driller with his helper making up to 300 or more metres of holes per shift. The holes are charged with ammonite and fired by means of an igniter cord. The salt is mucked with scrapers hauling up to 300 t of mineral per shift. The monthly advance of the stope in a room is 15-20 m. Explosives consumption amounts to 170-180 g per ton. The output per man in a mine is 5-6 tons or more per shift.

In room mining one should keep a constant watch over strict correlation of the size and disposition of rooms excavated at different floors (they should lie exactly one below the other) and also the size of intervening pillars. To compensate for the additional volume due to the loosening of the mineral the room should be undercut at a corresponding height prior to stoping. Sometimes undercutting is done with a slight advance, the salt being drawn through front through-cuts. Then the leading stopes of adjacent rooms must be pushed forward one somewhat ahead of the other.

Soft potassium salts are worked by the room method with rooms of smaller dimensions, greater size of pillars and stowing of the rooms after extraction. Substandard salt and chemical waste products are employed as mine-fill. In mining rock-salt deposits care should be taken to preclude penetration of water into the rooms.

Room-and-pillar and other mining methods. A distinct feature of the room-and-pillar methods as compared to room mining is partial recovery of coal from the pillars after the rooms have been worked out.

The conditions of application and the general sequence of mining by the room-and-pillar method have been described earlier with special reference to working medium-high seams. In the coal occurrences of the USSR room-and-pillar methods are rarely used because of considerable losses of the mineral and other limiting factors.

Apart from the above-discussed nonslicing methods of mining thick beds, use is sometimes made of systems involving extraction of coal in a series of horizontal or inclined slices with subsequent filling, similarly to mining ore deposits by cut-and-fill methods.

With these systems coal in the level is recovered in individual blocks, or without division into blocks, in one of the directions along the line of strike. Owing to small stope footage, difficulties of machine mining and high cost of operations these methods have not gained wide popularity in coal mining.

4-12. BRIEF INFORMATION ON NEW MINING METHODS. SEAM EXTRACTION IN DIFFICULT CONDITIONS

Combined Methods of Mining Inclined and Steep Thick Beds by Using Flexible Metal-Net Roofing

This novel system has been proposed by the Kuznetsk Coal Research Institute and successfully put to the test in a number of mines of the Kuznetsk coal basin. In this system a high seam is broken up into two slices: the top (assembly) slice 1.5 m thick and the bottom (main) slice, whose thickness equals that of the remaining portion of the seam (Fig. 126).

The top slice is worked out by long pillars on strike with caving or filling. The floor of the top slice is covered with a flexible metal roofing of special design (after the slice has been extracted throughout the length of caving interval of 6-8 m along the strike). Under this cover the bottom slice is extracted through sub-entries throughout the entire thickness of the slice, without supporting the stope area. The bottom slice is delineated at the elevation mark of the ventilating entry by drawing a transversely-inclined slice and laying down on its floor a flexible roofing like the one covering the floor of the top inclined slice. At the airway horizon the flexible roofing is fixed to the wall by means of suspension roof support.

Flexible metal roofing is made of band iron 50-100 mm wide and 3-3.2 mm thick. 50-mm bands are intertwined and laid down in both directions—along the strike and the dip of the bed (four or five bands per 1 m), 100-mm bands are so arranged that there are three bands per 1 m. Over the metal framework sheets of metal netting (25 × 25 mm mesh, wire diameter 2 mm) are laid down so as to overlap one another. The work in the second slice is not started until the stope of the upper slice has been advanced over a distance of 40 m.

Under the protection of the roofing the blasted coal passes by gravity through inclined cross-entries to the subjacent sub-entry along which it is delivered to the coal chute by conveyer. The hole rounds are drilled from a sublevel entry and inclined crosscuts.

The application of this system in coal mines of the Kuznetsk basin has greatly increased the efficiency of the facemen and cut the coal losses down to 12-15%.

With the use of flexible roofing a team of workers at the V. Vakhrushev mine in the Kuznetsk coal basin produced 38,372 tons of coal in 25 days (February-March 1967). The work was done in a block with a bed 12-13 m thick dipping at 30-32°, the inclined height of the level interval being 105-125 m. The level was divided into 3 sublevels. The seam was broken up into three slices across its thickness:

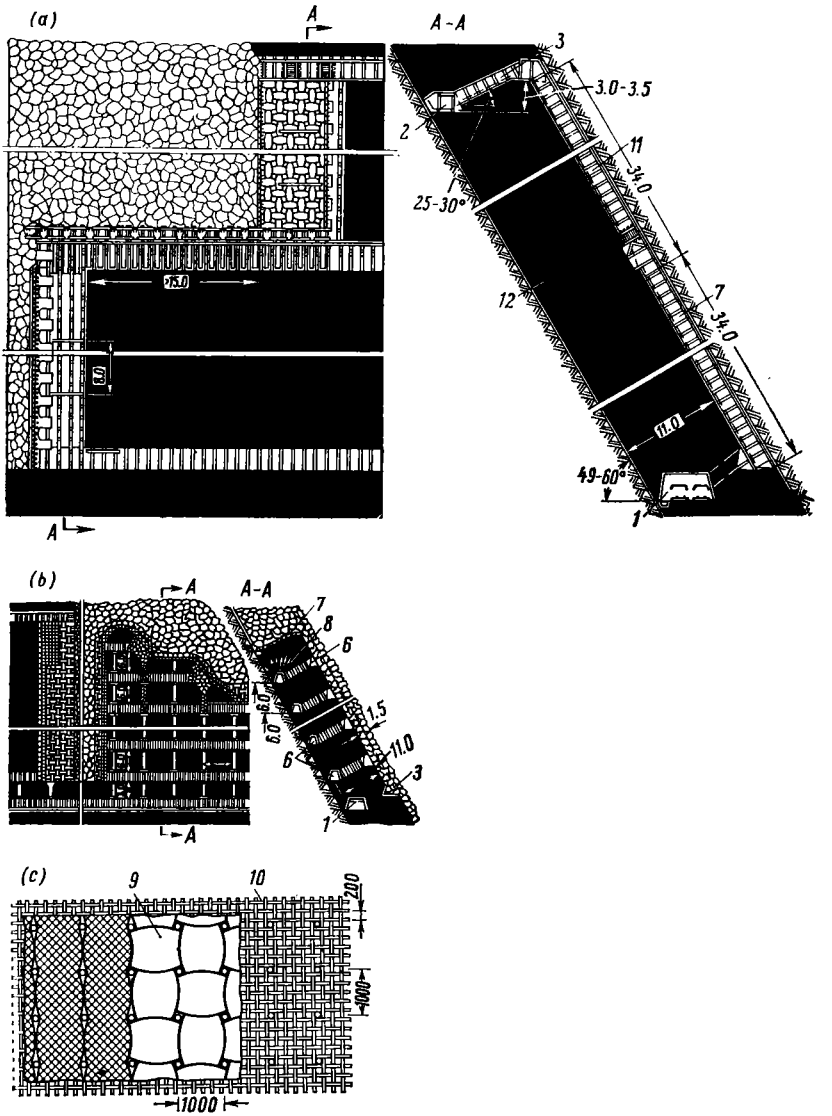


Fig. 126. Method of mining with division of the bed into two slices and the use of a flexible metal-net roofing between the slices (bed section across the strike):

a—at the stage of placing the roofing; *b*—at the stage of extracting main slice; *c*—flexible roofing; *1*—haulageway; *2*—ventilating entry; *3*—transversely-inclined slice; *4*—sub-entry; *5*—face entry; *6*—inclined crosscut; *7*—flexible roofing; *8*—blast-holes for caving the coal in sublevel; *9*—wire-netting No. 25; *10*—50 × 3 mm metal strip; *11*—top assembly slice (1.5 m high); *12*—main slice

the top (assembly) slice 1.5 m thick, the middle slice 6 m thick, and the bottom slice 5 m thick. The block was worked out in a series of pillars, each 20 m wide, with 4 coal chutes. Stoping was done down the dip simultaneously in two sublevels; with the working face of the superjacent sublevel having a lead equal to the width of the pillar. The maximum daily yield from a pillar comprised 2,243 tons, the output per faceman per shift being 91 tons. With work proceeding concurrently in 3 sublevels the output per working face can be brought up to 50-60 thousand tons per month.

Underground Hydraulic Mining of Coal Seams

In the USSR underground hydraulicking of coal was for the first time employed in the mines of the Kizel coal basin. Today the method of mining coal by means of hydraulic giants is successfully used in a number of mines of the Kuznetsk and Donets coal basins.

In 1966 hydraulicking in the USSR accounted for an output of 4.2 million tons of coal. In 1970 it is planned to produce 12 million tons of coal by this method.

Availing themselves of the experience gained by the Soviet Union, some other socialist countries have introduced this method in their mining industries.

The major advantage of coal hydraulicking consists in high labour efficiency. The method involves the use of hydraulic giants (monitors) with outlet nozzles ranging from 15 to 25 mm in diameter. In working high coal seams water discharge reaches 200-300 m^3/hr.

In hydraulically operated mines development workings are run under heads of 60-100 atm in beds containing coal of medium and sub-medium hardness, or strong but fissured coal (the advance rate reaches 5 m or more per one hour of work). In strong coal the head is raised and the solid coal in place is previously shock-blasted.

Hydraulicking of coal is used in flat, slightly dipping and steep beds.

Coal is extracted in retreating order from the boundaries of the producing section towards the shaft. As an example Fig. 127 shows mining by individual sublevels of a section-block 150 m long in a steep seam. Sublevels can be worked out both with and without flexible roofing. The sublevel interval is set at up to 10 m. Each sublevel is extracted by individual cuts under the protection of the overhanging coal canopy. Stoping is done simultaneously in several sublevels, the lower ones lagging behind. The broken coal is transported to the shaft along iron or glass troughs and/or tubes. Hydraulic pressure transport along tubes with entry gradients reduced to about 0.01-0.015 yields good results. With non-pressurized transport the adopted gradients are 0.04-0.05 for iron troughs and 0.02-0.03 for

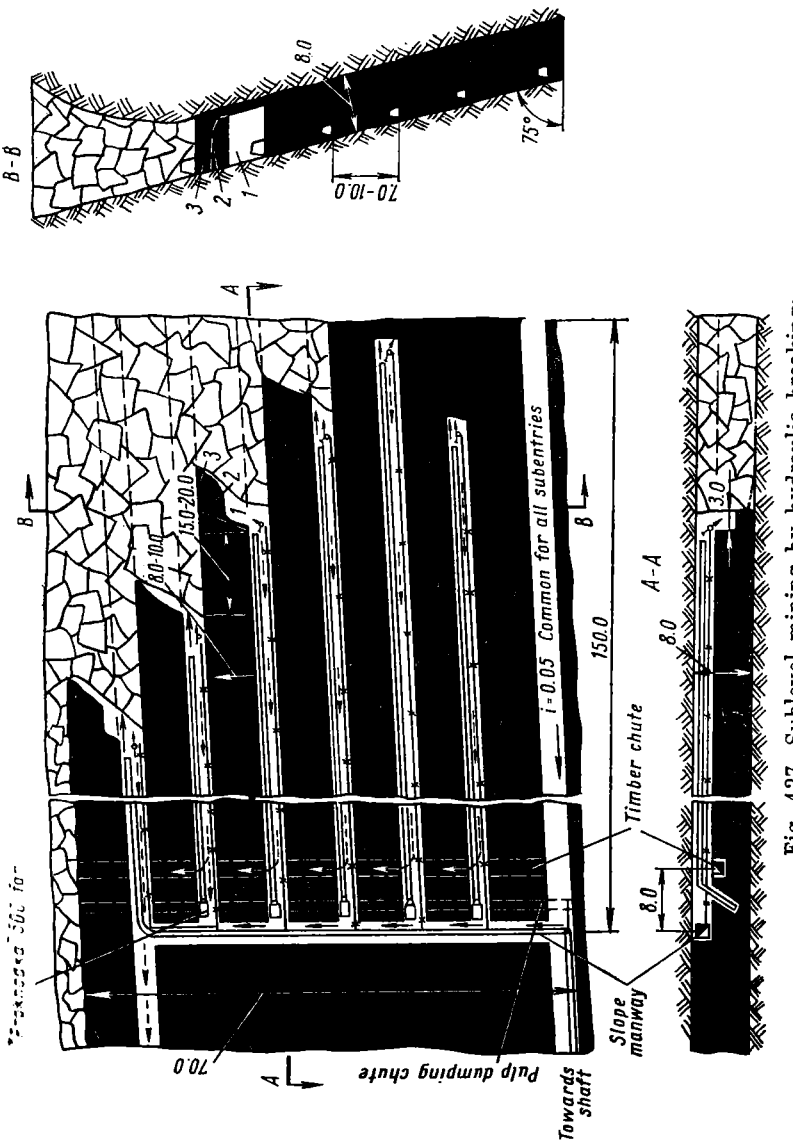


Fig. 127. Sublevel mining by hydraulic breaking:
1, 2, 3—sequence of coal breakage in the sublevel

glass ones. The solid-liquid ratio for the transport of broken coal is set at 1 : 3 to 1 : 4. In gently sloping beds long pillars to the rise are used with coal extracted down the dip.

The pulp delivered to the ground surface with the aid of high-pressure coal-sucking pumps is fed through a pipeline to a coal-concentrating mill, where the coal is dressed and dewatered. The clarified water is pumped back to the hydraulic giants.

Coal hydraulicking in the mines of the Kuznetsk coal basin has showed better results than the usual methods of coal mining in mechanized mines. The monthly output per man is 90-100 tons, electric power consumption per 1 ton of coal mined and dressed — 25-35 kW-hr, and timber consumption per 1,000 tons of coal mined — up to 15 cu m. The losses of coal have been considerably reduced in recent years. The reduction of coal losses is a task of prime importance at many hydraulically operated mines. One of the effective means towards this end is the use of flexible metal roofing.

Mining of Series of Contiguous Beds

Coal beds usually occur in series, or measures and are separated by interlayers of country rocks. Seams may be isolated or contiguous. Seams in a measure are considered to be isolated when they can be worked in any way or sequence. Beds are regarded as contiguous when their simultaneous mining is difficult or is unfeasible, or when the working of one of them complicates further mining of another.

The minimal thickness of an interlayer which under the conditions prevailing in the Donets coal basin permits undermining of the superjacent bed can be determined by the empiric formula deduced by A. Kilyachkov:

$$M = 12m + 3.5m^2, \text{ m}$$

where m is the extractable thickness of the subjacent bed in metres.

For other coal basins this formula should be corrected on the basis of field observations.

It should be borne in mind that in working steep seams both undermining from below and from above is possible, and such a possibility can be assessed by mapping out the corresponding areas of rock displacement (Fig. 128).

Undermining from above of the subjacent seam by the overlying one will take place when

$$M_1 < h \frac{\sin(\alpha - \beta_1)}{\sin \beta_1}$$

Undermining from below of the superjacent seam by the underlying one will occur when

$$M < h \frac{\sin(\alpha + \beta)}{\sin \beta}$$

where M and M_1 = distances between the seams;
h = level interval;
β and β_1 = angles of rock shift;
α = angle of dip.

The order of mining contiguous beds may vary. It may be descending, ascending, with separate and joint development, concurrent

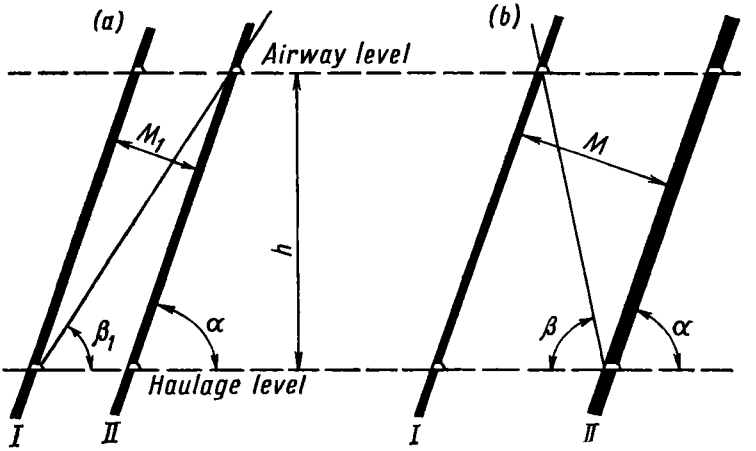


Fig. 128. Influence areas with a heavy pitch:
a—undermining from above; *b*—undermining from below

or consecutive extraction of contiguous beds. The choice of the mode of mining contiguous seams is influenced by:

the presence of seams in which there is a hazard of instantaneous outbursts of coal or gas, or of rock-burst shocks and their disposition in the series (undermining from above or from below of an unsafe bed diminishes or excludes the hazards);

the presence in the series of individual highly gas- or water-bearing seams (mining of a neighbouring bed facilitates degasification or dewatering of the seam);

the thickness and physico-mechanical properties of the interlayer rocks (when interlayer rocks are disrupted the ascending order of mining presents difficulties);

the presence of high steep seams (a thick bed should be placed at the top of the priority list of mining to ensure the attainment of the planned output of the mine);

susceptibility of the coal to spontaneous combustion (descending order of mining is recommended);

the mode of wall-rock control (complete caving does not favour the use of the ascending order of mining).

Account is also taken of other conditions, such as the angle of dip of the seams, the quality and strength of the coal, the distinctive features of the mining methods used, etc.

The descending order is the most popular in mining contiguous gently sloping seams. Joint development of contiguous beds is practised when expenses on driving and maintenance of development workings are less than or equal to the cost of separate development.

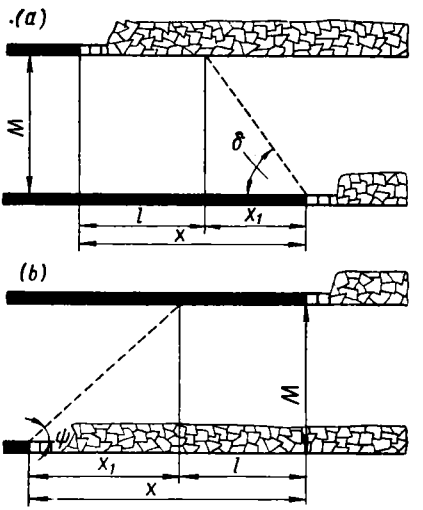


Fig. 129. Estimating reciprocal advance of working faces:
a—in downward mining; *b*—in upward mining

Simultaneous mining of contiguous beds is done in a variety of ways depending upon the thickness of the rocks forming the interlayer.

If the thickness of the country rock band is not great both seams (provided they slope gently) are worked out as a single seam (with joint or separate extraction). When the thickness of the country rock interlayer does not exceed 15 cm the rock is drawn together with the coal and hoisted to the ground surface.

In joint mining of seams the width of the interlayer should not exceed 0.7-1 m. Successive extraction of seams is practised when the thickness of interlayer rocks is in excess of 0.7-1 m.

In successive mining of beds the minimal advance of one working face over the other is estimated with an allowance for a reserve distance (Fig. 129).

In the descending order of mining

$$x = M \cot \delta + l, \text{ m}$$

in the ascending order of mining

$$x = M \cot \psi + l, \text{ m}$$

where M = the shortest distance between the beds, m;
δ, ψ = rock displacement angles;
l = reserve lead distance taken so as to equal the monthly advance of the working face (30-40 m).

The minimal advance thus established may prove insufficient, since the mining of one seam may affect the subsequent extraction of the other (dynamic shocks on abrupt subsidence and settlement of caved rocks). The time after which the working of the other bed can be started varies within a wide range and depends upon the thickness and physico-mechanical properties of the interlayer rocks, the extent of seam extraction, the firmness of the overlying rocks, etc. The rate of time advance is established by carrying out pertinent observations. It may be estimated approximately by using the following formulas (Fig. 129): in the descending order of mining: $t = \frac{1}{v}(M \cot \delta + l)$, months; in the ascending order of mining: $t = \frac{1}{v}(M \cot \psi + l)$, months, where v is the rate of advance of the production face per month in m. In mining contiguous seams no protective pillar should be left in the bed worked first (either with the ascending or descending order). Abandonment of protective pillars with the ascending order results in the bending or rupture of the top seam, whereas with the descending order areas of increased strains appear on extraction of the lower bed.

Mining of Seams Susceptible to Sudden Outbursts of Coal and Gas

Instantaneous outbursts of coal and gas occur in the majority of coalfields when mining is done at a depth exceeding 200-300 m. On sudden outbursts the coal is usually thrown off a considerable distance away from the working face. Outbursts are attended by the evolution of a large amount of gas. Beds presenting a hazard of instantaneous outbursts of coal and gas usually occur in strong enclosing country rocks, the mechanical strength of the coal being rather low. Geological disturbances favour sudden outbursts. The basic preventive measures in mining beds susceptible to instantaneous outbursts of coal and gas include:

preliminary degassing of an unsafe bed;
reduction of stresses in an unsafe bed;
increasing the mechanical strength of the coal;
building up of a mechanical obstacle to slow down the outburst in the initial stage.

The degassing of an unsafe seam and the reduction of stresses in it can be accomplished by advance mining of the neighbouring protective bed. The degassing of the seam can also be achieved by drilling boreholes.

The mechanical strength of coal can be raised during the degassing of the seam, and also through silicatization. Advance timbering

or shield support may be used as a mechanical obstacle to coal outbursts. A bed is considered to be protective when it lies at a distance of up to 50-60 m (along the normal) from the seam presenting hazards of sudden outbursts, unless experience has shown it to have protective properties at a greater distance.

Preliminary extraction of the protective bed makes it possible to simultaneously reduce the stresses in an unsafe seam and in its host country rocks, to increase the filtration capacity of the coal bed, intensify its degassing and increase the mechanical strength of the protected seam.

To drain the gas, boreholes 150-300 mm in diameter and 5-60 m deep (in individual instances 150 m deep) are drilled. In developing beds presenting an outburst hazard a minimum of openings are made in the seam. Development workings in the bed are run with preliminary stripping down of drainage boreholes, using advance protective support built up of metal tubes 4-8 m long. Such support may be combined with shock blasting (drilling and firing of a definite number of holes 1.5-2 m deep), the outburst of coal and gas occurring at the time of blasting. Seams presenting a hazard of coal and gas outbursts should be worked by continuous faces, using the level-longwall variant, which requires minimal excavation of main and subordinate development workings. The stope should be straight and continuous with the least possible number of starting slots. Development work and stoping should be conducted without abandonment of protective pillars. The entry heading forms part of the working face, being disposed on the same line. In mining a steeply dipping seam presenting an outburst hazard, breast-stoping with a continuous or stepped face and a minimum of benches is practised. Continuous working faces are made in stable enclosing country rocks when the coal is won by means of planers or mining combines, while underhand stoping is preferred in soft wall rocks when the coal is broken with pneumatic picks. In the latter case protective stages should be arranged over each bench.

During preliminary extraction of protective seams any method of rock pressure control may be employed in beds presenting a hazard of outbursts. In the absence of protective seams the best results are achieved by using rock pressure control with complete caving at minimal intervals.

Instantaneous outbursts are accompanied by the evolution of a considerable amount of methane. For this reason ventilating air currents emerging from unsafe sections should be directed along the shortest possible route to ventilation breakthroughs, preventing the ingress of these currents into workings where men are present. The scheme of separate ventilation of development headings and active stopes should be adopted, with the return current diverted

to superjacent mined-out floors. The aforesaid measures should be coupled with strict observance of general safety rules, such as the use of spark-proof machinery, rotary drilling of holes, provision of all miners with self-rescuers.

Mining of Seams Susceptible to Rock Burst Shocks

Changes in the stressed conditions of rocks due to the driving of development and productive workings, and the resulting increase in the bearing pressure lead to rock burst shocks.

Depending on their intensity and the conditions attending them rock bursts are accompanied by strong crackling sounds, violent outbursts or extrusion of coal and rocks, collapse of timbering in the mine-openings, a powerful air blast and, sometimes, the evolution of methane and rock tremor.

The sudden and instantaneous nature of rock burst shocks, which sometimes destroy mine openings over a considerable distance, and the absence, in many cases, of any warning signs make rock bursts one of the most dangerous phenomena encountered in mining operations. Rock bursts have long been known to happen both in the USSR and abroad. Particularly often they occur in deep mining. The frequency and intensity of rock bursts depend on the physical and mechanical properties of the coal and wall rocks, the mining depth, the thickness of the bed and back layers, and also on geological disturbances. Rock bursts occur mainly in firm ground and strong coal, for these are capable of withstanding substantial loads without destruction and can remain overstressed, thus accumulating a great amount of potential energy of elastic deformations.

In the Kizel coal basin rock burst shocks began to be felt at a depth of 200 m.

In mining beds subject to rock bursts approximately the same rules should be followed as in working seams susceptible to sudden outbursts of coal and gas. These include: preliminary extraction of protective seams; opening-up, development and mining by methods completely or partly ruling out abandonment of pillars and protruding edges of solid coal in place in the worked-out space. When protective pillars have to be abandoned they should be large-sized, their thickness being at least 25-30-fold that of the bed on each side of the working. Stone development of producing sections or panels is recommended. In mining a series of seams, group development with major openings made in country rocks or beds which present no rock burst hazard should be adopted. Low and medium-high seams should, as a rule, be extracted by longwall levels with advance mining and without leaving entry-chain pillars. In employing

pillar systems they should be designed so as to ensure the extraction of coal with a single-flank mine-field via a front gravity incline or slope without leaving stub pillars and with very insignificant advance of individual sublevels (5-10 m). Roof control is effected by complete caving or filling.

4-13. MAJOR TRENDS IN NEW TECHNOLOGY OF UNDERGROUND MINING OF COAL DEPOSITS WITH A MINIMUM OF WORKERS

In the course of the last 15-20 years great progress has been made in the USSR in the improvement of mining methods and the mechanization of industrial operations. Labour productivity should be increased and work conditions of the coal miners improved not only by perfecting the existing technology of coal production, but also by introducing new technology ensuring marked facilitation of the miners' labour and a rise in the overall production figures.

As shown by investigations conducted by Prof. A. Sudoplatov, Dr. Sc. (Eng.), from the Skochinsky Institute of Mining, one of the basic factors which enable the required number of miners to be greatly reduced is improvement in the rate of advance of the working face. This makes it possible to curtail not only the number of facemen, but also that of the auxiliary staff, which comprises a substantial proportion of the total number of workers in the mine. A sharp reduction in the auxiliary staff is one of the major methods of raising the overall efficiency per miner.

The problem of raising labour productivity in mines should be solved simultaneously with that of withdrawing workers from the production face (manless mining). The ways for attaining this goal include: new methods of mining requiring no presence of men at the working face, remote control of machinery employed in active working faces and the use of mobile support sets.

Among the machine mining methods requiring only a small number of workers (we shall call them manless mining for convenience) mention may be made of the following:

work with a supported face and the use of integrated mining units. This method of extraction has been described previously;

work with an unsupported face, such as hydraulicking, extraction by all-purpose mining machines (in short faces), by coal planers, saws; blasting, wedging, drilling and combined methods; the first three methods of mining with unsupported faces have been outlined in the foregoing text: below we give some brief information on coal extraction by mining combines in short faces, and also on the other methods, which have found little application so far.

Extraction by mining combines in short faces. This method is mainly employed in the USA, where wide use is made of self-pro-

pelled all-purpose mining machines and conveyers operating under extremely favourable mining and geological conditions, such as flat occurrence of beds, stable host rocks requiring no support.

The following results have been achieved in the coal mines of the USA, where room mining is largely used:

Number of active rooms in a panel	1
Total length of working faces, m	8
Number of workers	5
Advance of production faces per shift, m	20-22
Coal output per room per shift, tons	360-400

The output per man per shift in a panel is 70-80 tons.

Of late, in the rooms of American mines use has been made of remote control of stoping and transport equipment, which will enable production to be increased still further.

The blast-hole stoping method has been adopted from ore-mining practice. The breaking of coal in a heavily pitching seam is done by blasting deep holes. The broken coal is brought down to the entry. Stoping proceeds without timbering of the production faces.

Wedging is based on drilling holes with subsequent passing of a wedge through them. Thus the coal is detached from the solid mass in place. Steep seams with stable back and soft or cleavable coal favour the application of this method.

Stoping with the use of a mining sawing machine in a heavily pitching bed. The sawing machine is a plate 2.1 m long, 0.85 m wide and 5 cm high, made up of hinged sections. The plate carries cams with cutting bits fixed therein. The height of the starting slot is 7-9 cm. The sawing machine is pulled by ropes operated from a tugger hoist, which is mounted in the upper entry. The sawing machine moves along the breast of the face at a speed of 1 m/sec. The undercut coal is broken down by a breaking arrangement mounted on the body of the sawing machine.

Conditions favouring manless mining are listed in Table 15.

The blast-hole method of coal extraction may be employed in gently sloping beds (longhole-screw conveyer method) or in steep beds (blast-hole stoping). The major advantage of the longhole-screw conveyer method is the possibility of winning coal from extremely thin beds where mining by any other methods is very difficult, if not impossible. With this procedure labour efficiency is 2-2.5 times as high as in stoping with coal miners. The diameter of long blast-holes is chosen so as to conform to the bed thickness, their length being 30-35 m (with bilateral drilling the width of a long pillar reaches 60-70 m). The drilling set of the machine consists of a box and screw-conveyer links. The drilled-out coal is delivered into

Table 15

Conditions for Manless Mining

Method of manless mining	Conditions and field of application				
	Bed thickness, m	Dip angle, deg	Gas abundance	Coal strength	Properties of roof rocks
			In supported faces		
With self-loading hauling units	0.8-2.5	0-90	Any	Any	Readily caving and of medium cavability
			In unsupported faces		
Hydraulic	Over 0.8	5-90	Any	Up to hard coal	Above medium cavability
With coal miners (in short faces)	0.7-3.5 and more	Up to 17	Up to II category of fire damp abundance	Ditto	Any
With coal ploughs . . .	0.3-1.5	0-90	Any	Ditto	Ditto
With sawing machines	0.2-3.5 and more	45-90	Ditto	Ditto	Ditto
Blast-hole stoping . . .	0.4-1.6	0-90	Ditto	Ditto	Starting with readily cavable
Blasting	Over 1.5	45-90	Ditto	Ditto	Difficultly cavable
Wedging	0.7-1	45-90	Ditto	Up to medium strong	Ditto
Combined	0.4-1.6	45-90	Any	Any	Starting with medium cavability

Note: The table does not include information on the conditions of mining thick beds.

mine-cars or onto a conveyer spotted in the entry. Among the shortcomings of the method are high losses, up to 35-40% (mandatory abandonment of small pillars between long holes). These losses are, however, justified economically in mining very thin seams which cannot be extracted by any other methods. In the USSR sets of drilling equipment for working steeply pitching beds have been designed and are being tested in the field.

From automated working faces to automated mines. The prime objective for the immediate future is transition from automated production faces to automated mines. These will fully solve not only the economic problem, but also the social one—relieving man from hard physical labour. The miners will only have to go down into the mine periodically for assembly, inspection and repairs of the equipment, which will be controlled from the ground surface. In the USSR and in a number of foreign countries the construction of pilot automated mines has been started. Alongside automated mines, methods of manless extraction of minerals by chemical procedures are being worked out.

4-14. CHARACTERISTICS AND TECHNICO-ECONOMIC FEATURES OF BASIC METHODS IN MINING SHEET DEPOSITS

Coal occurrences in the USSR are mined by numerous methods, but some of them are used comparatively rarely.

Let us dwell upon the major systems of mining.

Conditions of Application of Basic Mining Methods In Working Coal Occurrences

1. Single-slice full-seam breast-stoping with extraction along the strike is employed chiefly in low and, more seldom, medium-high seams dipping at different angles, with firm and medium-firm wall rocks, and also in seams belonging to the super-category in respect to gas evolution and presenting a hazard of sudden outbursts of coal and gas.

2. Single-slice mining by long pillars with stoping along the strike is used predominantly in medium-thick beds with different angles of dip and wall rocks of varying strength, and also in low seams with unstable ground.

3. Mining in inclined descending slices with caving is used in high, gently sloping and inclined seams with enclosing country rocks of varying firmness. Each individual slice is extracted by continuous working faces (long-wall-level) or by long pillars on strike.

4. Mining in horizontal descending slices with filling is employed in predominantly steep thick beds. This method can be applied in beds dipping at any angles and with rocks of different stability.

5. Long-pillar mining down the dip with shield support (N. Chinakal's system) of ordinary design, without rollers, is employed in steep, high seams dipping at an angle of at least 55° and 4-10 m thick, with a regular mode of occurrence. When shield mining is impossible or difficult horizontal slicing is employed.

Design Features of Mining Methods

Single-slice methods with full-seam extraction (continuous and long-pillar stoping) in gently sloping and inclined beds.

Inclined level interval height, m	150-500 (usually 250-450)
Sublevel interval, m	100-150
Sublevel advance, m	10-20
Length of gravity incline, m	400-500
Average board run (cut depth), m	1.5-2.0 (with narrow-range extraction it is commonly 0.6-0.8)
Caving interval	Every 3-5 cuts (with mobile support, caving immediately after support shifting)

The softer the rock, the smaller the caving interval. With a tender, loose roof the caving interval is reduced to one cut. A smaller interval is also adopted with metal support.

Single-slice continuous face and long-pillar mining in steeply dipping beds.

Interval, m	
of level	100-150
of sublevel	50-75
Distance between slopes, m	100-300
Bench length, m	10-50 (usually 20-40)
Bench width	2-3 timber sets
Width of timber sets, m	0.8-1.0

Methods of mining thick beds

Inclined slicing-and-caving

Inclined level interval height, m	80-120
Slice height, m	2.8-3.5
Number of slices (depending upon thickness)	2-4

Horizontal slicing-and-filling

Level interval, m	40-100
Extraction section length, m	200-400

Slice height, m	2.8-3.5
Fill interval (run), m	4-6
Shield mining after N. Chinakal (sectionalized shields)	
Level interval, m	60-100
Width of pillar (block) on strike, m	24-30
Number of sections in the shield on strike	4-5
Number of sections in the shield across the strike	1-2
Section length on strike, m	6
Thickness of stump between pillars (blocks), m	1-2
Number of manways in pillar (block)	2
Number of coal chutes in section	1

Basic Technical and Economic Characteristics of Coal Mining Methods

These are listed in Table 16.

Efficiency characteristics of inclined slicing approximately correspond to those listed for the single-slice methods. Labour produ-

Table 16

Features	Mining methods		
	Single-slice continuous or by long pillars	Horizontal slicing (with air-blast stowing, timber support and conveyer haulage)	Shield mining (sectionalized shields)
Output per faceman per shift, tons	7-8	5-6	Up to 10-15
Coal losses incident to the method used, %%	10-15	5-7	Up to 25-30
Timber consumption (support), m^3/ton	0.04-0.05	0.06-0.08	Up to 0.02-0.03
Explosives consumption, g/ton	100-200	100-200	300-400
Metal consumption (depreciation of metal shield), g/ton	—	—	500-1,500
Average monthly advance of working face, m	35-40	Up to 50-75 (in one slice)	Up to 40 m of pillar down the dip
Comparative unit net cost of stoping, %%	100	110 120	60-70

Notes: 1. With the use of flexible integral (non-sectionalized) shields the efficiency of shield mining is higher (see above).
2. With manless mining efficiency of working coal deposits may be greatly increased.
3. With the use of mechanized integrated stoping units the average output per faceman per shift is 20 tons; the average monthly advance of the coal wall is 55 m and more, timber consumption being 0.006 m^3/ton.

ctivity in mining with thick slices may be assumed 20-25% higher than in single-slice methods. The overall mining cost of coal "free ground-surface bins", taken as 100%, is classified as follows according to the basic expenditure items: nominal wages plus additional wages and all extra charges 60-65%; supplies 10-14%; depreciation 12-15%; other expenses 10-18%. This percentage may vary somewhat depending upon mining conditions, methods of stoping. the extent of mechanization and the production capacity of the mine.

The labour productivity and face advance data listed in Table 16 do not represent ultimate figures and with further development of technology and improvement in the work organization should increase while coal losses will decline.

Measures for Further Increase in the Efficiency of Methods of Mining Coal Deposits

1. Extensive application of retreat mining in a level and of pillar systems in working beds of limited thickness.
2. Induced caving as the main type of roof control in winning gently sloping low and medium-high seams.
3. Machine stowing as the main mode of roof control in mining high steep seams substantially reducing coal losses and fire hazards.
4. Wide use of metal posts and mobile support in different mining conditions.
5. All-round mechanization of mucking coal onto conveyers. Machine loading of coal and broad utilization of coal miners and mechanized integrated sets in various mining and geological conditions are the next important steps in the mechanization of stoping operations and of development and solid work.
6. Broad introduction of belt and flight conveyers. Stream-flow conveyer transport should become the main type of haulage in producing sections. Designing of mining systems should be correlated with conveyer haulage and all-round mechanization of stoping operations.
7. Completion of change-over to remote control with automatic protection of mining combines, cutting, loading and other machines (manless mining at working faces).
8. Excavation of development and subsidiary development (first mining) openings by longholing.
9. Detailed and timely study of mining and geological features of occurrences, this being extremely important for appropriate selection and correct use of a particular system of mining.
10. Further study and establishment of the most effective methods of rock pressure control under different mining conditions.

11. Increasing the efficiency of the mine transport so as to fully utilize the productive potentialities of stoping which have in recent years augmented appreciably thanks to the introduction of complex mechanization, automation and better work organization.

12. Detailed and regular study and dissemination of methods used by individual progressive workers and entire teams and of improved methods of work organization, including all-round cross-flow planning.

4-15. METHODS OF UNDERGROUND PLACER MINING

General

The above-discussed continuous-face and pillar methods of coal mining are often employed in the working of other mineral occurrences.

Examples of pillar mining may be found in the working of phosphorite occurrences, placer deposits of gold and other minerals. Some basic facts about the mining of placer gold deposits by pillar methods are given below. As a rule, continuous-face systems are not applied in the mining of gold placers because of low rock stability.

Conditions of Mining Gold Placers and Systems Employed

Placers are sheetlike, very flat deposits of limited thickness (usually up to 2-2.5 m). In most cases the placer subterrane is firm bedrock. The capping represents loose mantle rocks of different stability, depending upon the composition, water-bearing capacity and climatic conditions.

In permafrost conditions the overlying rocks are quite firm and allow undercutting over extensive areas. The relative uniformity of conditions in placer mining restricts the number of possible and expedient systems of their working. Considering the poor stability of roof rocks under permafrost (as thawing progresses the back of the placer falls down), systems involving caving of the roof should be regarded as the best. In highly aquiferous placers drainage is recommended. In order to divert surface water a special outfall or water-diversion ditch is made on the ground surface and corresponding drainage facilities are provided.

The following methods are used in gold placering:

1. Long-pillar mining along the placer strike, with sand being extracted by a longwall and the mined-out space supported by pillars or by caving the overlying rocks.

2. Long-pillar mining across the strike, with sand drawn through pockets and subsequent caving of the capping, either with development in the bed itself, or by driving additional openings in the placer bedrock.

3. Pillar mining with the worked-out space supported by timbering and mine-fill.

The third group of systems are employed rarely (when a placer contains boulders used for filling) and therefore are not considered here.

Long-Pillar Mining on Strike with Sands Extracted by a Continuous Longwall and Subsequent Caving of Overlying Rocks

This system is used in mine-fields of considerable length under the following conditions: in solid permafrost ensuring sufficient

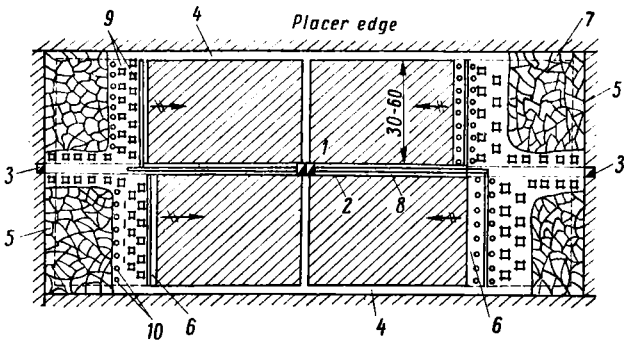


Fig. 130. Longwall pillaring in a placer:
1—shaft; *2*—main haulageway; *3*—air pit; *4*—fringe drifts; *5*—stope drift; *6*—continuous face; *7*—caved rock; *8*—conveyer; *9*—cribs; *10*—breaker timbering

stability of the wall rocks; with low water inflow; in placers at least 50-60 m wide (with a lesser placer width pillaring across the strike is employed).

Sand in the face is drawn either with or without preliminary thawing. Since the mined sands are washed during the warm season and because the washing of frozen sands has been introduced in recent years in a number of gold placer mines, preliminary thawing at the face has become unnecessary and for this reason mining of frozen sands without preliminary thawing is used more extensively at present.

Development and subsidiary development openings in long-pillar mining on strike (Fig. 130) include: haulage drifts run from the shaft up and down the valley (on the placer floor); fringe drifts along the

boundaries of the mine-field aimed at securing a second exit from the longwall during stoping operations; stope drifts from which stoping is started. Openings in the frozen ground are of normal section, without timbering, they are pushed forward at a high rate. The length of the longwall (width of the pillar) is taken to be 10-12 to 30 m and more, depending upon the physical condition of the sand, strength of the back, and intensity of stoping. In mining frozen sands the length of the wall may reach 30 m and more. When the sand is worked with preliminary thawing the wall should be comparatively short. Frozen sand is usually extracted starting from the mine-field boundaries towards the shaft along the entire breast of the stope, directly from the stope drift. To break down frozen sands in the wall drilling and blasting is employed.

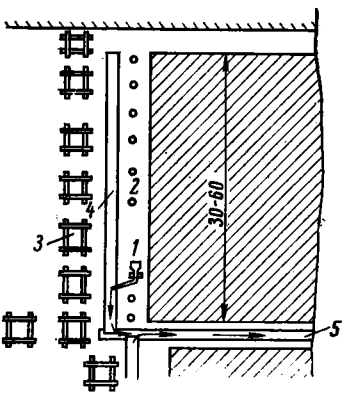


Fig. 131. Mucking machine in longwall placering:
1—loader; *2*—props; *3*—cribs; *4*—face conveyer; *5*—drift conveyer

This method is simpler and more effective than sand extraction with preliminary thawing. The holes should be at least 1.5 m deep. The height of the stope should equal that of the minable portion of the placer, but should not be less than 1.4 m. The operation cycle in the stope includes drilling, blasting, loading and hauling of the sand to the drift. The loosened sand is slushed to the drift or loaded with small-size mucking machines (Fig. 131) into mine-cars or onto conveyers. In this case the sand is delivered from the stope area to the main drift by short belt conveyers and from there onward along the drift by continuous conveyer lines up to 100-200 m long each. When sand is broken down by blasting the roof is supported by temporary pillars (Fig. 132). The haulage drift is maintained by leaving similar pillars. As the longwall moves forward temporary pillars are recovered with subsequent induced caving of the roof. Temporary pillars are of rectangular or square shape, their size can be determined by using the formula proposed by Academician L. Shevyakov, account being taken of the compression strength of frozen ground (35-40 kg/cm^2).

Until 1941 mining of sands with preliminary thawing had been widely used. It is more complicated and requires reinforced timbering of the stope with cribs and temporary face timbering.

With a sufficiently stable back, timbering is done by setting up rows of cribs spaced at a distance of 2-3 m, while rows of posts are put up along the face of the stope. To induce caving, some of the cribs are removed. Breaker timbering is sometimes installed along

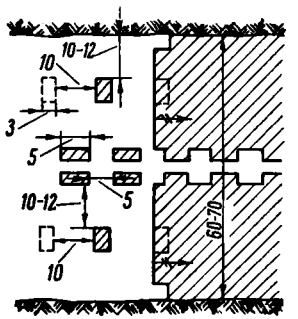


Fig. 132. Temporary pillars in permafrost placering

the future caving line. After thawing the sand is drawn with mechanical devices—pneumatic hammers and shovels, then loaded and hauled in the same way as frozen sand.

In the steam-processing of sands the ground is thawed out during two shifts, then broken down and transported from the stope area in the third shift.

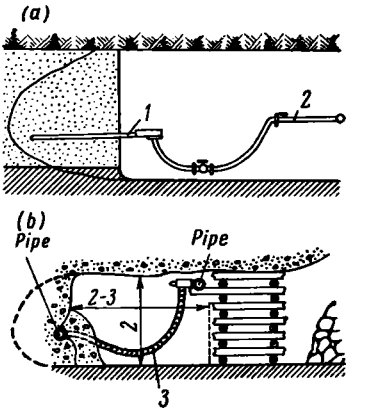


Fig. 133. Methods for thawing frozen ground:
a—thawing by drill-steel steam points; *b*—thawing with pipes; *1*—steam point; *2*—steam pipes; *3*—rubber hose

Cribs are shifted simultaneously with the thawing out of the sand, and preparation for thawing is timed with the breaking down and delivery of the sand.

The thawing of frozen sands in the stope can be done by passing steam either through special hollow drill-steel (so-called points) or through pipes (Fig. 133).

In the former case use is made of hollow drill-steel 25 mm in diameter and 2-3 m long, one end of which has a pyramidal head and the other a flat steel cap welded to it. The drill-steel is first rammed into the ground to a depth of 15-20 cm, following which steam is passed through it; as the sand is thawed out the drill-steel is driven with a hammer into the ground to a depth of 1.5-2 m, this operation taking 2-6 hours. Then the drill-steel is withdrawn and the breast of the stope is treated with steam passed through pipes in abundant quantities.

The stope is steam-processed for 6-10 hours, depending upon the type of ground. Thereafter, it "sweats" for 10-24 hours. Should it prove difficult to drive in hollow drill-steel, especially in the presence of large-sized clastic material, the other procedure of steam-thawing is employed. A cut is made at a height of one-third of the stope face, into which pipes are inserted. The pipes are then covered with sand, which is thoroughly compacted. The pipes employed for this purpose are 2-4 m long and 18-50 mm in diameter, both their ends are plugged. The pipes have 2-3 mm holes perforated in them (three holes per 1 metre of the pipe). Steam is supplied to the pipe through a rubber hose. The laying of pipes takes 3-4 hours, direct "steaming" 6-7 hours and "sweating" 8-9 hours. Steam boilers with a heating surface of 12-30 sq m are set up on the ground surface, as close as possible to the site of steam-thawing; the steam pipelines from the boiler plant are often laid in a specially sunk borehole, near the projected site of work. One steam-point requires 1-1.5 sq m of boiler heating surface. Steam pressure in the boiler is 6 kg/cm^2. Steam may also be supplied from electrically heated boilers installed underground.

The stope advance rate with thawing by means of hollow drill-steel is 1.5-1.7 m per cycle, and through pipes 0.5-1 m. Steam consumption per 1 cu m of thawed-out sand equals 30-80 kg. Output per faceman per shift amounts to 3-4 cu m and more.

In mining frozen sands with irregular or small width of the placer use may be made of long-pillar mining across the strike with extraction of the sand by a longwall and caving of overlying ground.

Long-Pillar Mining Across the Strike with Sand Extraction by Splitting

Variant with Productive Development. The method is mostly used in mining thawed placers. In the last 10-15 years it has supplanted the formerly popular pillar-and-bord system.

The basic conditions for the application of long-pillar mining (single-road stall method) with caving of the roof in the worked-out

space are: the thawed state of the placer and peats, sufficient width of the placer which permits cutting up pillars of adequate size; moderate water inflow; thickness of the auriferous bed not exceeding 2.5-3 m; regular relief of the bedrock.

The development openings include: the main haulage and offtake drifts, parallel fringe and extraction drifts driven from the main drift to both fringes of the placer and spaced at 10-15 m. The distance between the extraction drifts corresponds to the width of the pillar. The length of the extraction drift is taken to be 30 m at the most; with a greater placer width additional parallel drifts are run.

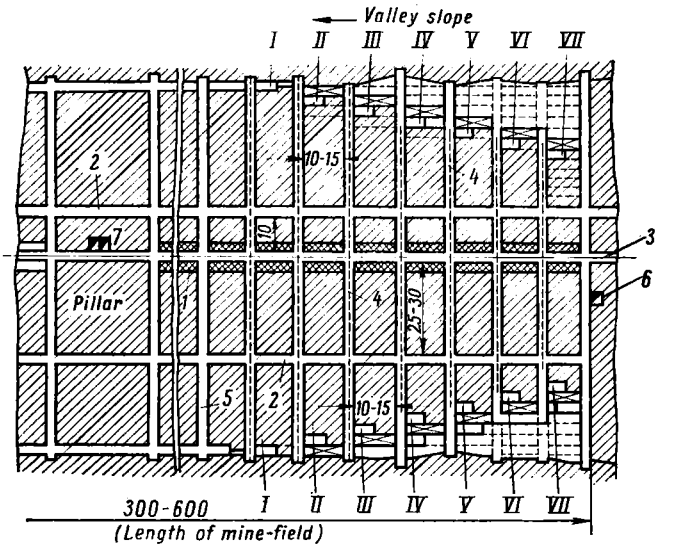


Fig. 134. Stoping in placer mining by long pillars across the strike:
1—waste packs; *2*—service drifts; *3*—main haulageway; *4*—conveyer; *5*—extraction drift; *6*—auxiliary shaft; *7*—hoisting shaft; *I, II, III, IV, V, VI, VII*—minable pockets

The relative positions of the above-listed openings in the mine-field are shown in Fig. 134. The main haulage drift, serving at the same time as an offtake drift, is driven a few tens of metres ahead, so as to drain the sections intended for sand extraction. The main drift should run along the thalweg of the placer. The driving of the main drift, which is usually done in moist sand, is an extremely difficult job and for this reason the drift is often supported with timber sheet piling. Permanent timbering consists of close timber sets reinforced by relief posts. For better maintenance of the main drift (after working out of the mine-field this drift is often used

as an offtake drift in working a neighbouring mine-field) it is additionally supported by waste packs with timber spacers on both sides of the drift (Fig. 135).

Stoping in the mine-field progresses from its boundaries to the placer edges. Long pillars are recovered through individual pockets 3.3 m wide excavated from the corresponding drifts (Fig. 134).

Not more than three pockets should be worked simultaneously in one pillar, one of the pockets being excavated, another undergoing

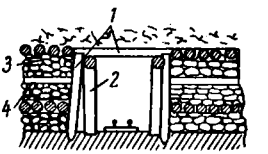


Fig. 135. Main drift timbering:
1—timber set; *2*—relief post; *3*—waste pack; *4*—wooden spacer

"cleaning up" and the third either being prepared for caving or actually caved. If the roof is firm two pockets may be made concurrently, their extraction then proceeding simultaneously from two sides by converging stopes.

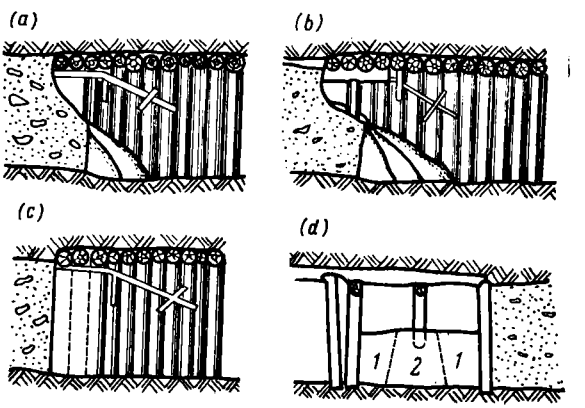


Fig. 136. Sequence of sand extraction and timbering in a pocket of a placer with weak ground:
a—mining of sand from top portion of stope; *b*—setting up of helper post; *c*—setting up of main posts; *d*—sequence of drawing of sands from side and centre areas of stope; *1*, *2*—slices

Sands are drawn in the pocket with pneumatic hammers and shovels and, when necessary, blasted out. As it moves forward the pocket is immediately supported by closely spaced timber sets with relief posts. To speed up the installation of timber sets in the stope

area, a toe or top cut is made. With a top cut, cross-beams are suspended by iron bars from previously installed timber sets (Fig. 136).

With a toe cut, posts are inserted into the hitches, after which the sand is quickly drawn from the stope and other members of the timber set are added. Once the sand is recovered from the pocket the bedrock is cleaned up (it is often enriched in metal down to a depth of 10-20 cm). The mined-out pockets are caved as the stoping progresses, the support being previously crushed, with up to 30-60% of the mine-timber pulled out. With a strong and dry back and regular placer bedrock the pillars may be worked out by longwalls running over the entire width of the pillar. The width of the pillar should not then exceed 10-13 m. If the back of the placer is not sufficiently drained the roof in the pockets cannot be allowed to fall, for this may cause silt-ground inrushes. In these conditions

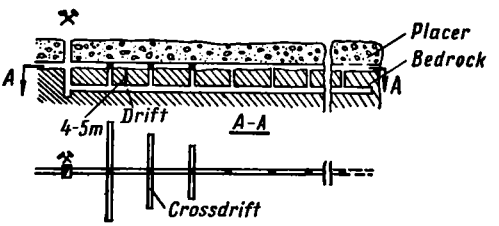


Fig. 137. Disposition of mine openings in placer bedrock

the pillars may be mined with partial filling of the worked-out space and erection of waste-pack walls and cribs.

Variant Involving Drivage of an Additional Drift in the Placer Bedrock. This modification is employed in highly aquiferous placer beds where excavation of mine-openings is extremely difficult.

In such conditions it is expedient to run the main drift in the underlying bedrock; this requires extending the shaft into the subterrane so that a protective pillar not less than 4-5 m wide is left when pushing forward a drift at a reasonable gradient in the bedrock (Fig. 137). Leaving a pillar of a greater size is not advisable, since this would worsen drainage conditions and augment the amount of development work in connecting the drift with the bed. In broad placers transverse openings are driven additionally from the bedrock drift. Experience shows that this drift should be run 100-150 m ahead of the openings in the placer bed so as to ensure its proper drainage.

After dewatering the work in the bed is carried on in a manner similar to the main variant of the system. The present modification appreciably facilitates and cheapens the mining of sands from highly aquiferous beds.

New Technological Trends in Underground Placer Mining

These include:

(1) hydraulicking of sands with their subsequent transportation by conveyers to the ground surface or to special suction dredges delivering pulp to the surface;

(2) extensive use of metal posts and mobile metal support in mining of sands;

(3) change-over from automation of individual operations to all-round automation of all mining operations, including extraction, transport and washing of sands.

CHAPTER V

METHODS OF UNDERGROUND MINING OF ORE DEPOSITS

5-1. FUNDAMENTALS OF MINING ORE DEPOSITS

Distinctive Features of Working Ore Deposits

Most of the ore deposits represent lenses, stocks, domes, pockets, lodes or a combination of ore bodies. The mode of occurrence is extremely irregular, the angle of dip and the line of strike being liable to frequent variations. It is especially difficult to mine lode deposits representing a set of thin veins with irregular spatial position and outline. In working such deposits extensive prospecting is needed, and its results should be brought in line with the exploitation requirements. A specific feature of ore deposits is the diversity of the properties of the ore and host rocks and the complex composition of the ore. Both the ores and enclosing country rocks vary from friable to very strong, from soft to extremely stable. The complexity of composition is typical primarily of non-ferrous ore deposits which are usually polymetallic. These ores often belong to distinct grades and classes which need different methods of dressing and processing (rich and lean, sulphide and oxidized, copper and zinc-copper, chalcopyrite and pyrites, etc.).

By reason of high technological standards and the necessity of delivering ores of different grades and classes, selective extraction in mining ore deposits is often a matter of prime importance. In working very thin ore bodies, separate recovery of the ore and barren host rocks, picking and sorting of gangue in underground workings and on the ground surface acquire great importance.

Because of high demand for raw materials used in smelting ferrous and non-ferrous metals ore deposits and placers in the USSR should be worked with utmost intensity. In 1967 a total of 168 million tons of commercial iron ore (about 250 million tons of run-of-the-mine ore) were produced in the USSR. The overall bulk mined in 1966 by the enterprises of non-ferrous metallurgy comprised 505 million cu m, the planned figure for 1970 being about 700 million cu m. Underground mining accounted for approximately 30% of iron and 35-40% of non-ferrous ores.

Basic Mining Characteristics

In establishing factors influencing the choice of a mining method and in deciding other issues pertinent to the working of ore deposits one should be guided by the accepted practical characteristics of ore bodies, the ore and enclosing country rocks.

Classification of Ore Bodies According to the Dip Angle. As regards the angle of dip, ore bodies are classified as *horizontal* and *conventionally horizontal* with a dip angle of 0-3° (the maximum angle of 3° is adopted in conformity with the maximum permissible track gradient for electric locomotive haulage of the muck directly from the stope area); *gently sloping* ore bodies with a dip angle of 3-30° (the ore in the stope area is hauled by mechanical devices—scrapers, conveyers); *inclined* ore bodies with dip angles of 30-45° (the ore in the stope area can be hauled by simple arrangements, such as metal troughs and metal-lined chutes); *steep* ore bodies with dip angles of 45-90° (the broken ore can move in the stope area by gravity).

Classification of Ore Bodies by Thickness. As concerns their thickness, the ore bodies are classed as *very thin* (up to 0.7-0.8 m thick), *thin* (0.7-0.8 to 2 m), *medium-thick* (2 to 5 m); *thick* (5 to 15-20 m), and *very thick* (over 15-20 m). The classification of ore bodies according to their thickness is based on the most frequently employed methods of excavating development openings, or on the direction of stoping. Thus, in very thin ore bodies stoping and excavation of major development workings (drifts, raises) are done with additional ripping of wall rocks; in thin ore bodies stoping is carried on without ripping, while major development workings are driven with ripping of wall rocks; in ore bodies of medium thickness the stoping and excavation of major development openings are effected without ripping of wall rocks; in mining thick ore bodies stoping is usually done along the strike of the ore body; very thick ore bodies are worked across the strike.

Strength of Ores and Host Rocks. The hardness of ores and enclosing country rocks can be determined by using the simplified classification of rocks according to their minability proposed by Prof. B. Boky. The rocks are divided into five groups: loose, soft, friable, strong, and very strong.

For more precise assessment of the hardness of the ores and host rocks the hardness scale of Prof. M. Protodyakonov is used. The rock hardness figures established according to this scale may, whenever necessary, be verified by compression tests of rock specimens of definite shape and size. The International Bureau of Soil Mechanics in Prague recommended in 1961 that test specimens be obtained by drilling out a core with a height and diameter of 40-45 mm, the

parallelism and flatness of its faces being strictly observed. The Bureau established a scale that includes 12 classes of rock strength from 30 to 3,000 kg/cm^2.

In making calculations for drilling and blasting work and in setting up the relevant standard characteristics one should use special classifications of rocks in respect to drillability and blastability. Such classifications were first proposed by Prof. A. Sukhanov, Dr. Sc. (Eng.). Today they are widely used in a number of branches of the Soviet mining industry. More detailed information on the classification of rocks as to their hardness, drillability and blastability may be found in textbooks on drilling and blasting.

In solving problems of rock pressure, drilling of the ground and other problems relating to the mining of mineral deposits, great importance is now attached to the classification of the physical properties of rocks. A classification of physical properties and the corresponding standard nomenclature of rocks have been suggested by V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR. The practical classification of physical properties of rocks recommended by V. Rzhevsky is given in Table 17.

Table 17

Classification of Physical Properties of Rocks

Groups of parameters	Basic properties	Designations
1. Density	Volume weight	γ
	Porosity	p
2. Mechanical	Compression strength	σ_{comp}
	Tensile strength	σ_{tens}
	Young (elasticity) modulus	E
	Poisson ratio	μ
3. Thermal	Heat conductivity	λ
	Specific heat	c
	Coefficient of linear expansion	β
4. Electromagnetic	Electric resistivity	ρ
	Permittivity	ε
	Magnetic permeability	μ'

Stability of Ore and Enclosing Country Rocks. By stability is meant the ability of rocks not to collapse when exposed from below and on the sides. The stability of rocks greatly influences the choice

of a mining system. A classification of the stability of ores and enclosing country rocks is presented below.

1. *Soft and very unstable ground not allowing even the slightest exposures in the roof and on the walls of mine-openings.* Advance timbering is often necessary when making openings in such ground. Soft grounds include loose, running and friable aqueous grounds, for example, water-abundant chlorite-sericitic schists. In selecting mining systems and the sequence of working individual portions of a deposit it is very important to know whether such ground is present in the hanging or foot walls of the occurrence, since special precautionary measures should then be envisaged.

2. *Unstable ground permitting very limited exposures of the roof and walls in the opening immediately adjoining the working face.* In such grounds the back and walls of the opening should be supported right after its excavation.

3. *Medium-stability rocks allowing limited exposures of the roof and walls in the opening.* The need for artificial support of exposed ground does not arise at once, but with the passage of time and not always over the entire exposure area.

4. *Stable rocks permitting extensive exposures of the roof and particularly of the walls, and requiring support only at individual spots or at certain intervals.*

5. *Very stable rocks allowing very large exposures of the roof and walls without any artificial support.*

Although this classification lacks a quantitative index, it allows a practical approach to the choice of a mining system taking into account the stability of the ground.

In appraising the ground with regard to its stability it is very important to ascertain whether cavings occur instantaneously or gradually, over small or extensive areas, in lumps, large blocks or entire layers, whether there are any external signs, such as cracks, increased pressure on the timbering, etc. It is necessary to determine the effect of air and water on the stability of the ground after a certain lapse of time. We should also emphasize the importance of establishing the actual exposure areas in the roof and walls of mine-openings (in square metres) in the absence of support or with certain types of timbering or filling in active mines. The practice of mining ore deposits makes it possible even now to set the maximum allowable exposure areas in the roof or walls at 200 sq m in medium-stability rocks, 500 sq m in stable rocks and 1,000 sq m and more in very stable rocks. In working ore occurrences stable exposures of 2 thousand sq m and more have been encountered.

In the future, it will be possible to ascertain permissible exposure areas with a greater degree of accuracy and to find a practical procedure for rock control in any given mining conditions. This should

be done by using systematically processed observation data and establishing the basic physico-mechanical properties of rocks and the trends in the variation of rock pressure.

Particular attention of workers of the geological-prospecting and mine-surveying services should be drawn to the determination of the stability of rocks. In estimating the stability and strength of rocks consideration should be given to tectonic dislocations and jointing, which often predetermine these characteristics. Details of rock jointing are established by using special circular diagrams taking into account the number, orientation and size of joints. When excavating mine openings and sinking boreholes it is necessary to collect information allowing jointing to be characterized with the use of such diagrams. The effect of jointing on the stability of solid rock in place can be assessed by optical modelling or by modelling on equivalent materials. This method will help define the effect of the nature, size and orientation of joints upon the stability of rock in place.

In practice hardness and stability often do not coincide. Thus, hard ore may be unstable, and stable ore may not be strong.

When selecting a mining system, a number of other factors should be considered in addition to the features listed above. These factors are enumerated in Sec. 5-10.

The wrong or insufficiently accurate definition of the characteristics of the working of a deposit results in improper selection of the mining system. Geologists play an important part in establishing factors influencing the choice of mining methods.

It is not always possible to present a full characteristic of an entire deposit, but the mining and geological features of its sections to be worked first should be as complete and clear-cut as possible. A mining and geological characteristic of an ore body and its host rocks enables one to define more accurately the technical and economic parameters of mining methods.

Methods of Supporting a Mined-Out Space

For safe and convenient conduct of stoping operations the worked-out space must be supported artificially. This is done by means of ore pillars, timbering, temporary abandonment of broken ore in the worked-out area, or filling it with stowing material (waste, slag, etc.). The selected method of support depends on the stability of the ore and wall rocks. With ore and wall rocks susceptible to caving the mined-out space is often supported only at the working face (active stope area), whereas the remaining space is filled with caved rock.

Ore Pillars as Support of the Worked-Out Space. Permanent ore pillars are left to support the mined-out area when extracting ore bodies containing stable ores of low value with stable enclosing country rocks. Sometimes this method is employed in conjunction

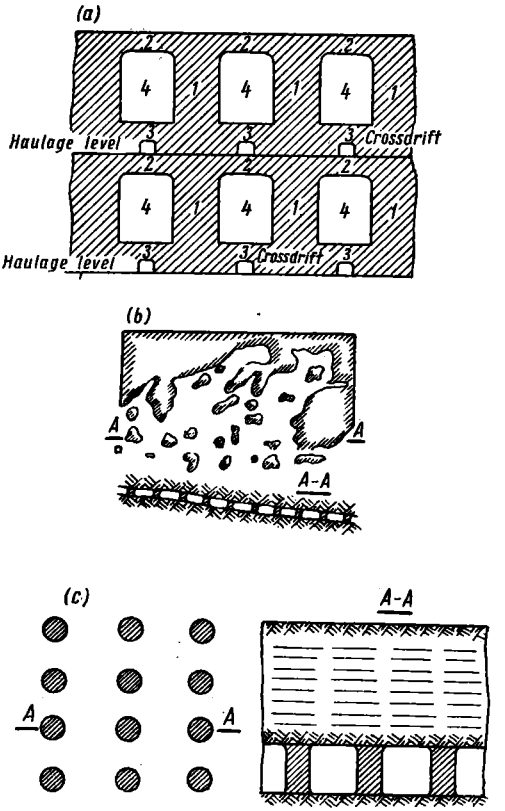


Fig. 138. Schematic representation of pillars:
in multilevel mining of a thick ore body (section along the strike): *1*—intervening pillars; cap pillars; *3*—sill pillars; *4*—stoped-out rooms; *b*—casual pillars; *c*—regularly arranged pillars

with other types of support. Usually temporary pillars are left, to be robbed at a later date. The following types of protective ore pillars are distinguished: intervening, cap, sill and stope (left inside the stoping area) pillars (Fig. 138).

Stope pillars are commonly used for supporting the back in mining flat and gently dipping ore bodies. They may be arranged in a casual manner (with frequent inclusions of barren rock or poor ore) or

regularly, and should have a definite size and shape. Casual natural pillars are not robbed, while regularly arranged pillars are partially recovered during second mining, observing the prescribed precautionary measures.

The size of pillars is usually determined on the basis of practical experience. There is so far no generally accepted procedure for estimating the size of support pillars or of their spacing.

A procedure for estimating the size of an intervening pillar as proposed by Academician L. Shevyakov has been described above.

To calculate approximately the size of natural support pillars one must have the following basic data: physico-mechanical characteristics of the useful mineral (ultimate compression and tensile strength, etc.), geological section of the overlying rocks (up to the ground surface), physico-mechanical characteristics of the capping; volume weight of the useful mineral and barren rocks.

Of the methods for calculating natural support pillars the one suggested by Academician L. Shevyakov is the most reliable. It is, however, applicable only to flat deposits with a limited mining depth. No well-founded methods for calculating support pillars at great mining depths are available so far. A method for approximate estimation of the pillar size with special reference to inclined ore bodies has been proposed by S. Borisenko, Dr. Sc. (Eng.). It is based on the rock pressure borne by an intervening pillar whose volume equals that of a creeping hanging wall prism. It is assumed that the whole load is borne by intervening pillars alone, no account being taken of the load partially transferred to the cap pillar and the filling in the room which slightly raises the margin of safety. The pillar is analysed for permissible stress with an allowance made for a safety factor of 2 or 3.

Among the latest methods of estimating the size of natural support pillars in mining at great depth the method suggested by I. Rivkin deserves attention. It is based on the maximum equivalent rock exposure span and functional design characteristics.

In determining the size of support pillars of adequate strength other methods may also be employed, such as centrifugal modelling, modelling with the aid of equivalent materials, the optical method, and experimental-field observations.

At a number of Soviet mines good use is made of the microseismic method for measuring stress conditions of solid rocks in situ which enables a tentative assessment of the stability of rock exposures to be made. Special piezoelectric devices are employed to this end.

The microseismic method utilizes the ability of rocks to reproduce sounds under the effect of pressure. The sounds become audible before the pressure becomes destructive. With rising pressure the frequency of repetition of the sounds increases differently in diffe-

rent rocks. Knowing the sound frequency increment in a particular rock one can establish permissible minimal dimensions of support pillars. Feeble sounds developing in rocks are picked up and recorded by special highly sensitive receiving and amplifying devices.

The microseismic method is applicable only in strong rocks, such as quartzite and other rocks of similar type.

Broken Ore as Support for Mined-Out Space. The worked-out area may be supported with broken ore provisionally left therein. This, however, can be done only where the enclosing country rock is so strong that it does not collapse on discharge of the broken ore.

Mine-Fill as Support for Stoped-Out Area. To support the worked-out space either filling alone, or filling in combination with timbering (in unsteady ore and host rock) is employed. In working ore deposits waste left in the mine is often used as stowage material. This waste comes from ripping and slashing of the enclosing country rock when mining thin and very thin ore bodies, when sorting out the ore in the stope area (if solid ore contains gangue intercalations and inclusions), when making permanent and development mine-openings in country rocks; from special underground workings, where mine-fill waste is excavated and then passed down from the superjacent worked-out levels (transfer of mine-fill is permissible when caving of host country rocks and of the ground surface is possible and also when mining fire-safe ores).

When using stowage material passed down from the ground surface the mine-fill may include: waste dumps on the ground surface, granulated slag derived from smelters, concentration-mill tailings, and barren ground excavated in open-pits. Tailings are used after the separation of the sludge. Of late, self-cementing fill has found wide use in mines of the USSR. The techniques employed in mine filling have been described earlier.

Timbering of the Worked-Out Space. This method of support is employed independently only in mining ore bodies of limited thickness with firm and stable host rocks. In the practice of mining ore deposits, especially non-ferrous metals, timbering is commonly done concurrently with stowage or with subsequent caving of the enclosing country rocks.

The following types of stope timbering are used, depending upon the design of the support (Fig. 139): (1) simple stull timbering in ore bodies up to 5 m thick (usually up to 3 m); if the stull sets are arranged over the drift and filling is used reinforced stulls are put up; (2) support with individual upright posts and stull-caps or back slab-lagging, this is used in horizontal or slightly dipping ore bodies and with limited rock pressure; (3) combined timbering using vertical and horizontal posts with or without lagging of walls and the back of the stope (composite stull-sets); used in ore bodies

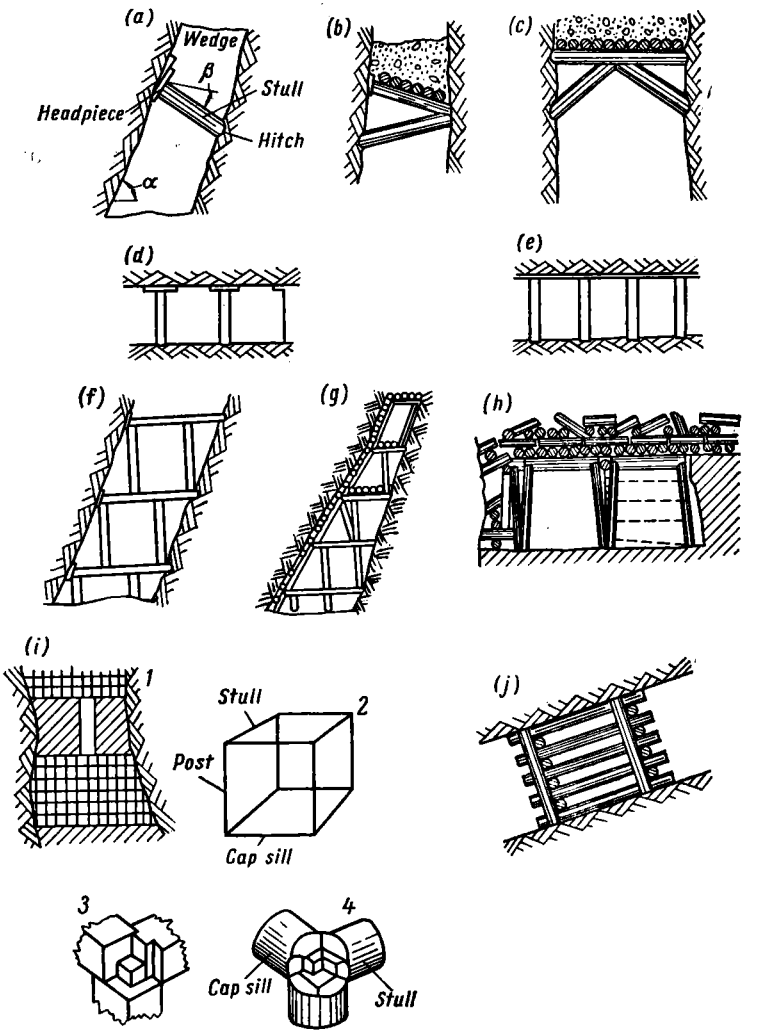


Fig. 139. Types of stope support in use:

a—support with anchor props (simple stull timbering); *b*—support with main and helper anchor posts (reinforced stull); *c*—support with main and battered anchor props (reinforced stull); *d*—support with posts and headpieces; *e*—support with posts and head slab-lagging; *f*—combined support with upright and horizontal posts without wall lagging; *g*—same with lagging of the hanging wall; *h*—three-stick timbering; *i*—square-set timbering (*1*—general pattern of timber setting; *2*—square-set members; *3,4*—timbering key); *j*—crib

3-5 m thick and with higher rock pressures; (4) three-stick timbering (close-set, or spaced at intervals); used in working ore bodies of limited thickness, and less frequently in thick ore bodies with intensive rock pressure; (5) square-set timbering comprising upright posts and horizontal stulls; the butt ends of cap sills, posts and stulls are shaped so as to make a tenon joint with square or rectangular tenons; the general view of the timbering, square-set members and key is shown in Fig. 139; this type of timbering is mostly used in mining thick ore bodies, in combination with filling; (6) cribbing; used in working ore deposits, usually not as an independent type of timbering, but rather as auxiliary support in cases of increased rock pressure.

Information on the setting up of various types of timbering is presented below along with the description of the corresponding mining methods.

Timber is the basic material employed in making mine support used in ore deposits.

In post-war years extensive experiments with the use of metal posts and prefabricated reinforced concrete support were carried out successfully in a number of mines. Today, experimental research on the use of mobile shield support is underway in some mines and suspension-roof support is being introduced on a large scale.

Methods of Stoping in Relation to the Outlines of the Working Faces

In the practice of mining ore deposits diverse shapes of the stoping front are adopted, depending upon the particular mining and geological conditions and requirements for efficient, convenient and safe breaking of the mined bulk. The following methods of stoping are distinguished depending on the outline of the working faces (Fig. 140): overhand, underhand, and breast-stoping.

Overhand Stoping. In overhand stoping the ore is broken by benches in ascending order, the lower benches leading the upper ones. The miners work beneath the horizontal or sloping plane of the bench. Overhand stoping may be done by horizontal, inclined or diagonal slicing. The adoption of a particular version of overhand stoping is contingent upon the angle of dip of the ore body, the mode of support used in the worked-out space, and the ore properties. The presence of joints and their direction in the ore may prove essential. The latter factor should be taken into account for safety reasons.

Overhand stoping by horizontal slices is used most frequently.

With an angle of dip exceeding 45-50° overhand stoping allows the mineral to be moved by gravity.

Underhand Stoping. This is a mode of mining in which the stope is made in the shape of benches disposed in descending order, with advanced stoping of the upper benches.

When drilling, the miner stands on the solid ore at the toe of the bench. Holes are drilled downwards, except for reliever holes, which may be run horizontally or slightly upwards. In underhand stoping getting the ore down requires some reshovelling.

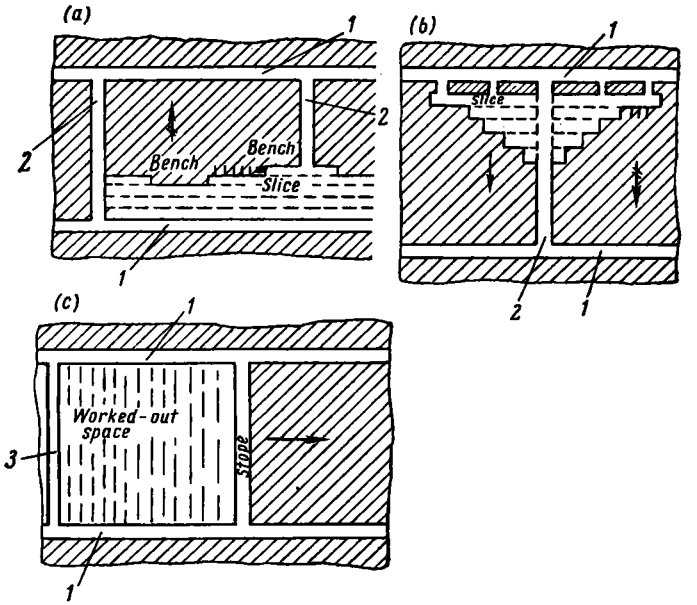


Fig. 140. Stoping in blocks depending upon the shape of the working front: *a*—overhand stoping by horizontal slices in a steep ore body (section along the strike); *b*—underhand stoping by horizontal slices in a steep ore body (section along the strike); *c*—breast-stoping in a flat ore body (plan view); arrow indicates general direction of ore extraction; *1*—drift; *2*—raise; *3*—breakthrough

Breast-Stoping. In this mode of extraction the stoping operation front runs along a single line. This line may extend in any direction, depending upon the geological conditions and the design characteristics of the mining method.

Breast-stoping is commonly used in horizontal or slightly sloping ore bodies. The conditions of application, the advantages and disadvantages of the basic modifications of stoping are described when discussing the various mining systems.

Stoping Based on Cyclic Operation

Cycle and Cyclic Organization of Stoping in Mining Ore Deposits. Ore recovery in the stope area consists of separate mining operations. The major operations are: drilling and blasting, mucking of ore and waste rock, timbering, filling of the goaf or caving of enclosing country rocks. All other work performed in the stope area, such as delivery of equipment and materials to the working face, extending of pipes, track laying, etc., is of an ancillary nature. All stoping operations are closely related to one another and should be strictly coordinated. The whole process is usually initiated by drilling, this being followed by other regularly recurring stoping operations. Some of them may be combined. Ancillary work is usually done simultaneously with the basic operations.

The stoping cycle in mining ore deposits is a regularly recurring complete series of mining operations required for moving the active stope area in a definite direction and over a definite distance in conformity with the conditions governing the conduct of work with the adopted mining method, for instance, moving the back of a slice by the height of one slice in cut-and-fill methods of mining, etc.

Cyclic-Work Schedule and Its Compilation. The principle underlying cyclic stoping was described in detail when discussing the mining of sheet (coal) occurrences.

Standard charts of the organization of stoping operations used in ore mining usually contain a brief characteristic of the prevailing conditions of mining and of the available equipment, a cyclic-work schedule, labour distribution chart, and technical and economic data.

Dilution and Losses of Useful Mineral

Exploitation of mineral wealth in the USSR is carried on with the highest possible quantitative and qualitative recovery of the mineral reserves.

During the last 30 years a great deal of research work has been done in the USSR with a view to establishing the level and structure of mineral losses in the mining industry.

This work made it possible to take practical steps aimed at reducing losses and dilution of ores in the working of mineral deposits, particularly of ore bodies. Below we present basic information on the estimation and recording of losses and dilution and measures to reduce them.

Basic Concepts and Terms. Today the term *dilution* is used to designate two essentially different phenomena in the working of

mineral occurrences. On the one hand, the word dilution, as used in the mining literature, denotes the reduction in the content of useful constituents in the extracted ore as compared to their proportion in the mass of ore in place. Accordingly, the *dilution factor* is the ratio between the reduction in the content of the useful constituents after dilution to their initial content in the ore in place. On the other hand, according to the "Unified Instruction" now in force at mining enterprises, the dilution factor characterizes the proportion of barren host rocks contained in the extracted material. By *losses* is meant the decrease in the amount of useful constituents in the extracted mineral as against their amount in the solid mass in situ. Accordingly, the ratio between the lost amount of useful components and their reserves in the earth's crust is called *the coefficient of losses.* Losses may also be characterized by the *coefficient of recovery* from the earth's crust, i.e. the ratio between the extracted amount of the useful constituents to their reserves in the earth's crust.

With the use of correct methods for mining ore deposits the dilution factor usually varies from 0.05 to 0.20 and the coefficient of recovery from 0.95 to 0.80, depending upon the mining, geological and technico-economic conditions and the adopted mining method (in some instances losses and dilution reach still higher figures [1]).

The Effects of Losses and Dilution. Maximal extraction of a mineral from the earth's crust without dilution with barren country rock is one of the major requirements in the working of mineral deposits. Losses and dilution of a mineral substantially reduce the efficiency of a mine. Losses diminish the extractable reserves of a valuable mineral.

Losses of reserves also increase the expenditures on exploratory, capital and development work per unit of the extracted mineral. The loss of broken ore causes economic damage because of unproductive expenses incurred in breaking the lost ore.

Dilution lowers the quality of the mineral, raises the cost of concentration, milling, metallurgical processing, transport and handling, especially when long-distance hauls are involved. In many cases it brings down the yield of useful constituents at concentration plants and mills.

All this emphasizes the importance of measures aimed at reducing losses and dilution of ores in the working of mineral deposits, particularly those containing valuable ores which require complicated technological processing.

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[1] In bulk mining of narrow veins dilution may be as high as 70-80%, and in room mining of salt losses reach 40-50%.

Types of Losses. So far there is no unified and approved classification of mineral losses. In the available instructions for the recording of losses in individual branches of the mining industry the losses are divided into the following broad categories: (1) mineral losses associated with the mining, geological and hydrogeological conditions (this group includes losses in pillars and benches of the mineral abandoned to protect mine workings against water inrushes from underground reservoirs or quicksands; losses in areas featuring considerable tectonic dislocations of beds, deposits or lodes, etc.); (2) mineral lost in safety and barrier pillars (safety pillars are intended to protect shafts, surface structures and plants, water bodies, streams and rivers, towns and settlements, while barrier pillars serve to separate the mineral reserves of a mine or a group of mines from the worked-out reserves or those to be extracted in other mines or combined mining enterprises); (3) exploitation losses, which include losses of mineral in the course of mining. This type of losses chiefly depends upon the mining and geological conditions, the adopted method of working a deposit and a consistent application thereof. The exploitation losses may be subdivided into losses of broken and unbroken ore. The first subgroup includes: losses in stowing, losses on the walls and the floor of the stoping area, losses due to the mixing of ore with barren rock, losses in delivery, haulage and storage (the last-named kind of losses is sometimes recorded separately as an independent group of losses in transportation); the losses of unbroken ore include: losses in pillars left during stoping and losses owing to incomplete extraction along the perimeter of an ore body.

All the above-enumerated groups of losses are recorded separately. Mineral reserves belonging to the first and second groups may be written off as lost irretrievably only with permission of the State Mining Inspection Authority. As compared with other kinds of losses the exploitation losses are more important as a rule, since they occur regularly and are quite considerable. The percentage of permissible exploitation losses is fixed in the mining project for a particular occurrence. The figures for such losses are given below together with the description of the relevant mining methods. Correct recording of exploitation losses and implementation of measures designed to reduce them are of great practical importance. Particular attention should be paid to the reduction of losses whose proportional share is the highest: losses in stowing, due to mixing of ore with barren rock and in abandoned pillars.

Types of Dilution. Dilution largely depends on the mining conditions, the system of working employed and the proper conduct of operations.

The mining of deposits with weak host rocks and indistinct contacts is attended by appreciable dilution of the ore. Some methods,

such as block and sublevel caving, are characterized by high dilution, and so does the extraction of thin and very thin ore bodies because of the forced breaking of dead rock to obtain a stoping area of minimal breadth.

Dilution is mainly caused by gangue getting into the ore during its breaking; mixing of the broken ore with the filling material, exfoliating or caving gangue or low-grade ore.

Estimation and Recording of Losses and Dilution. Overall exploitation losses of the useful constituent in per cent can be estimated by using the formula:

$$P = 100 - E,$$

$$E = 100 \frac{T_a (C_a - C_r)}{T_p (C_p - C_r)} \%$$

where: P = exploitation losses of the useful constituent, %;
E = extraction of the useful constituent, %;
T_a = the amount of actually mined ore, tons;
T_p = the amount of solid ore in place to be extracted, tons;
C_a = actual content of the useful constituent in the extracted ore, %;
C_p = the content of the useful constituent in the solid ore in place according to the assay, %;
C_r = the content of the useful constituent in the rock contaminating the ore, %. With $C_r = 0$ the formula assumes a simpler form.

The losses of the useful constituent calculated by the above formula characterize the losses of the mineral as well. It should be borne in mind that it is wrong to calculate ore losses by taking the ratio between the tonnage of the extracted bulk and its reserves in situ, since the excavated bulk will partly contain barren rock and therefore the extraction figures will be overstated.

The total dilution of the mineral during exploitation (in per cent) can be determined by the formula

$$R = 100 \frac{C_p - C_a}{C_p - C_r} \%$$

where R is the dilution of the mineral in %. With $C_r = 0$ the formula becomes simpler.

From the above formula it follows that the dilution of ore is largely dependent upon the content of metal in the rocks contaminating it. Therefore high dilution figures do not always mean low extraction efficiency. For instance, if the content of iron in the solid ore in place is 55%, in the excavated ore 52% and in the overly-

ing rocks 0% the dilution will amount to 5.5%, whereas with the content of iron in the superincumbent rock comprising 45% the dilution will equal 30%. By pulling the ore over a longer period of time its losses in the latter case will be reduced by about 50 per cent. Consequently, although the dilution of the ore in the former case is approximately 5.5 times as low as in the latter case the recovery figures for both metal and ore will be much lower.

The dilution factor expressed in fractions of unity can be determined as follows:

$$D_f = 0.01R$$

The percentage content of metal in the extracted bulk at certain dilution factors and content of the useful constituent in the solid ore and enclosing rocks can be calculated thus:

$$C_a = C_p - D_f (C_p - C_r)$$

If the ore is sorted the percentage content of metal in the picked (salable) ore can be calculated by the formula

$$C_s = \frac{C_a - (1 - V_s) C_{pr}}{V_s}$$

where C_a = actual content of metal in the extracted ore (the bulk to be picked), %;
C_s = content of metal in the salable ore, %;
C_{pr} = content of metal in the picked rock, %;
V_s = yield of salable ore per 1 ton of the ore to be picked, tons.

The recording of ore losses and dilution at enterprises is done by using the established accounting forms (monthly and quarterly forms for recording exploitation losses and dilution of ore for operation blocks, the mine and each individual method of mining). Exploitation losses are recorded separately for broken and unbroken ore. One should bear in mind that in a mining method involving multistage working of blocks correct judgement as to the losses and dilution of the ore cannot be formed until data on the extraction of the entire block have been received, since monthly reports would not be representative enough.

In the final assessment of the indirect method for estimating losses and dilution as described above it should be admitted that this is an approximate procedure, for possible errors in the initial data may greatly exceed the values sought for. This refers both to the content of metal in the contaminating rocks and in the ore lost. In the method under review the contents of the useful constituent in the positive reserves and in the ore lost are taken to be equal, whereas in some instances, especially in mining lode deposits,

the lost fine ore contains a higher percentage of the useful constituent.

Special investigations carried out at the Skochinsky Institute of Mining under the guidance of M. Agoshkov, Corresponding Member of the Academy of Sciences of the USSR, have shown that for some mining methods the direct procedure is to be considered a more accurate way of determining losses and dilution (they are estimated on the basis of direct measurements of the amount of ore and metal actually lost and of the volume of admixed host rock and excavated bulk).

The direct method is particularly efficient in working intricately shaped ore occurrences, when ore reserves subject to extraction and the content of the useful constituent in them cannot be predicted with any reasonable degree of accuracy. In such instances the indirect method cannot be applied. In the direct method of recording the basic types of losses in individual blocks must be reported at the mine and broken down into the following items:

losses due to incomplete extraction along the contours of the ore body;

losses in stowing;

losses due to mixing of ore with caved rock;

losses in pillars and avalanches;

losses in transportation from the stopes to discharge points on the ground surface;

losses in stockpiling on the ground surface;

losses in sorting.

In the direct method the overall losses of metal in per cent are estimated by the formula

$$P = \frac{(P_1 + P_2 + P_3 + \ldots + P_n)}{(T_a + P)} 100\%,$$

where P_1, P_2, etc. = individual items of metal losses, tons or kg;
T_a = actual metal yield, tons or kg;
P = total losses of metal, tons or kg.

The direct method of determining dilution by measuring the volume of the admixed host rock and the extracted mineral is in the stage of elaboration. At the Skochinsky Institute of Mining other, more precise and simple procedures for estimating dilution have been proposed, of which the method to be used in mining thin ore bodies suggested by A. Nazarchik deserves special attention. The method obviates the necessity of determining the content of metal in railroad cars entailing labour-consuming assaying with inaccurate results.

A. Nazarchik suggests estimating the dilution factor (the weight ratio between the mined ore and the vein matter contained therein)

by using the formula

$$R_0 = \frac{(m_s - m_v)\,\gamma_r + m_v\gamma_v}{m_v\gamma_v}$$

or

$$R_0 = \frac{[(m_s - m_v)\,\gamma_r + m_v\gamma_v]\,C_v}{(m_s - m_v)\,\gamma_r C_r + m_v\gamma_v C_v}$$

where m_s = width of the stope area, m;
m_v = mean width of the vein, m;
γ_r = volume weight of the host rock, tons/m^3;
γ_v = volume weight of the vein matter, tons/m^3;
C_v and C_r = content of metal in the vein matter and host rocks, respectively, %.

The first formula is employed when the enclosing rock does not contain any metal, the second when it does.

To make an accurate estimate of dilution in the delivered ore one should also take into account any possible losses of the vein matter during its discharge and in the fill.

In working lode deposits the major steps aimed at reducing dilution include: the adoption of systems involving separate extraction of ore, drilling of small-diameter holes with 28-30 mm bits, increasing the intensity of stoping and the use of suspension roof support to avoid exfoliation of the rock, and also extensive sorting of the lode stuff (additional general measures will be listed below).

One can make the following general conclusion with regard to recording losses and dilution, based on the research carried out in the USSR.

In mining lode deposits where losses include ore with a higher proportion of useful constituents than in the case of workable positive reserves the errors in calculating the coefficient of losses and the dilution factor by the indirect method exceed those made with the direct method. The direct method allows block-wise recording and timely detection of the channels of ore losses so as to take preventive measures. The indirect method does not offer this possibility as a rule.

Establishing Individual Characteristics for Estimating Losses and Dilution of Ore. Accurate recording of losses and dilution becomes possible only when the relevant characteristics are determined correctly and in good time.

Brief information on the determination of individual characteristics is given below (more detailed data may be found in special manuals on geological subjects).

Content of metal in solid ore. This is determined during major and subsidiary development and extraction of ore blocks by assaying with the use of pertinent geological documentation. The assay

results are plotted on a plan and this serves as a basis for estimating the mean content of metal in a block. Where interlayers of gangue are present in the ore body they should be taken into account. If such interlayers are fairly thick and can be separated during breaking or picked thereafter they should not be included in the individual stope samples. Otherwise interlayers should be included in the sample and considered as natural contamination.

The quantity of metal in solid ore. Along with estimating the content of metal in the solid ore of a deposit its quantity should also be determined, i.e. a well-founded estimation of its reserves made. The accurate determination of ore losses largely depends on the correct estimation of reserves. In calculating the amount of ore in the solid mass particular attention should be given to the correct estimation of its moisture content and volume weight.

Average content of metal in the mined ore bulk. The ore delivered to the surface is assayed by grab sampling from mine-cars. For checking by the bulk method every tenth, twentieth, etc. mine-car is sampled. Samples from cars are taken either by the grid sampling procedure or by the so-called "envelope" method, from five points, each sample weighing 0.2-0.5 kg and more. In large mine-cars samples are taken from six to eight points. Prior to assaying, the samples are ground and their size reduced.

Sometimes samples are taken when the ore is discharged onto a picking belt, unloaded into concentration mill bins, or fed from the bins to the mill for processing.

Tonnage of mined ore. The weight of the ore extracted at a mine is determined by weighing when shipping it to the consumer.

Recording of the mined ore separately for blocks and producing sections should, as a rule, be done by weighing it in mine-cars on the ground surface and directly in the main underground levels.

The results of mine-car records of extracted ore tonnages should be checked at least once a month. This is done by using mine-surveyor's measurements of the remaining ore stocks in stockpiles and bins and by keeping track of the weight of ore shipments.

At all mines records by weight are matched by records of the extracted ore according to mine-surveyor's measurements of the worked-out space and the data of geological documentation.

General measures aimed at reducing exploitation losses and dilution of ore.

1. One of the major measures to prevent losses and dilution of ore is the proper choice of the mining system with reference to the existing mining and geological conditions, and also the correct application of the adopted method. Thus, in mining with supported stopes and fill it is important to ensure the proper framing, setting and wedging of the timbering to prevent rock inrushes from the wall

and roof, tight lagging of ore and waste chutes, good flooring on the mine-fill prior to breaking of ore, etc.

2. The correct sequence of working the levels, horizons, producing sections and blocks of the ore body with regular mine-surveyor's measurements and quality control of the ore (regular sampling of the ore by stopes and commercial sampling according to specially elaborated patterns).

3. Regular and timely cleaning of stopes with the broken ore; keeping scaffolds, flooring, dumping chutes, draw holes, drag paths and raises in good shape, thorough clearing of ore from the abandoned goaf.

4. Intensive advance of stoping and development faces and timely packing of the worked-out space. With a slow advance of faces losses and dilution of ore are always higher because of exfoliation of the host rock. To ensure intensive extraction in producing sections and systematic reclamation of levels one should avoid unnecessary scattering of workplaces over numerous areas and levels of the mine.

5. Establishing well-founded quality standards and minimal commercial content of the useful constituents in the ore mined in each mine.

6. Increasing the responsibility of the engineers, technicians and workers and stimulating their interest in the optimal qualitative and quantitative results of mining operations.

Classification of Methods for Mining Ore Deposits and the Sequence of Their Study

Ore deposits are characterized by extremely great variety and complexity of their working, therefore the number of methods and their variants used in the practice of mining ore occurrences is quite considerable (over 150). The diversity of mining conditions and the great number of existing systems complicate the elaboration of a simple and harmonious classification of methods for mining ore deposits.

The high requirements imposed on such a classification (it should reflect the essence of the methods and serve as a basis for their designation, selection and study) make it still more difficult to set up a single classification for systems employed in mining ore deposits. A great number of such classifications based upon different principles have been devised, but none of them fully answers the purpose.

Most of the Soviet mining specialists believe that a classification of mining methods should be based on the methods of supporting the stope area, or the condition of the latter. There is no doubt that

the mode of supporting the stope area and also its condition are the major features characterizing a mining method, since they determine the general design of the system and its technical and economic characteristics. These features, however, do not fully reflect the peculiarities of the systems, for instance they fail to reflect the general sequence of mining and the associated development work.

The mode of supporting a stope area or its condition would be the appropriate basis for establishing classes of mining methods; the general sequence of extraction, which in many instances determines the amount and orientation of development work, should serve as a basis for groups of systems; finally, the features characterizing the details of stoping (methods of breaking off the ore, the shape of working faces, the type of support, etc.) should be taken as a basis for setting up subgroups of mining systems.

Soviet mining specialists have introduced a number of amendments into classifications based upon the mode of support of a stope area or its condition. Among the latest classifications the one proposed by M. Agoshkov deserves special attention.

This classification with modifications concerning individual groups and subgroups of mining methods (made by the author of this book) is largely adopted in the present text. In the classification given below the classes of systems are designated by Roman numerals I-VIII, their groups by Arabic numerals 1, 2, 3, 4 (subgroups of the methods, by letters a, b, c, d, e).

When using this classification it should be borne in mind that the designation of the first group of systems "with block mining" signifies that here the whole ore body or level is divided into separate blocks extracted over the entire area without abandoning any pillars (Fig. 141*a*).

The designation of the second group of methods "with room mining and subsequent recovery of pillars" means that this group is characterized by the extraction of rooms with abandonment of temporary pillars, usually recovered in second mining (Fig. 141*b*).

The designation of the third group of methods "with continuous breast-stoping of the ore" signifies that in this instance the ore body or level is not broken up into blocks, rooms and pillars and stoping proceeds in one or several definite directions [from centre to flanks, from flanks to centre, or simultaneously over the entire area of the ore body (Fig. 141*c*)].

The designation of the fourth group "with room-and-pillar mining" means that here rooms are extracted and permanent pillars—stumps—abandoned (Fig. 141*d*). In grouping up the mining systems one should bear in mind that blocks and rooms may be worked out with or without dividing them into sublevels and this influences

the design features of a method, the general sequence of extraction, and the scope of development and subsidiary development work.

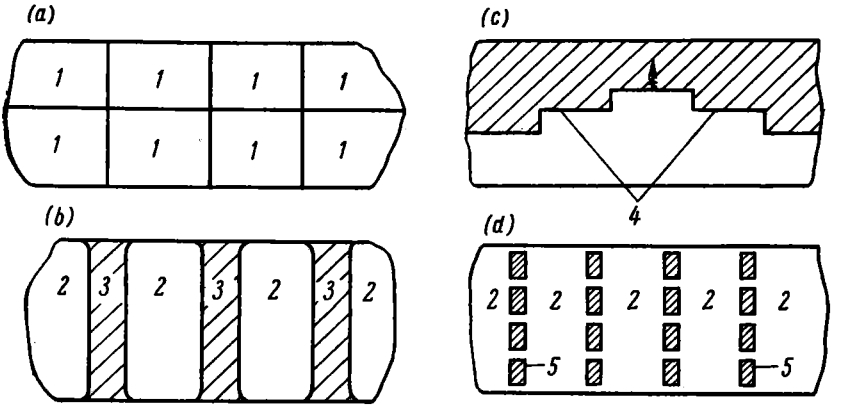


Fig. 141. Diagrams illustrating determination of the groups of mining methods (ore bodies in plan):
1—block; *2*—room; *3*—intervening pillar; *4*—breast-stoping front line; *5*—pillar

The classification (Table 18) includes frequently employed methods only. Other systems can be readily introduced into the classification proceeding from the above-stated principles of dividing the mining methods into classes and groups.

Table 18

Abridged Classification of Methods for Mining Ore Deposits

Class	Designation of Classes of Mining Methods	Frequently employed groups of methods
I	Open stope methods	1, 2, 3, 4
II	Shrinkage stoping methods	1, 2, 3
III	Cut-and-fill methods of mining . .	1, 2
IV	Methods with artificial support of the stope area	1
V	Methods with artificial support and filling of the stope area	1, 2, 3
VI	Methods of mining with caving of host rocks	1
VII	Methods of mining with caving of the ore and host rocks	1
VIII	Combined methods	2

The explanation of the conventional numerical designations of the groups of mining methods is as follows:

1—group of methods with block mining;
2—group of methods with room mining and subsequent recovery of pillars;
3—group of methods with continuous breast-stoping of the ore;
4—group of methods with room-and-pillar mining.

A more detailed classification of methods for mining ore deposits with indication of their subgroups is presented later along with the description of methods in each individual class.

In the classification given above the class of combined mining systems includes methods involving extraction of rooms by one method and pillar recovery by another. More detailed explanations relating to combined methods are presented below together with the description of the VIII class of methods.

A factor of paramount importance in classifying the mining methods is the setting up of a single scientific classification of methods for mining deposits of solid minerals (coal, ores, nonmetallic minerals). Prior to accomplishing this difficult and extensive work it is necessary to improve the existing separate classifications of methods used in the mining of sheet, ore and other occurrences. Such an improvement will appreciably facilitate devising a single, unified classification.

Among the unified classifications available at present in the USSR those worthy of attention are the general classification of the methods for underground mining of mineral deposits (G. Tsulukidze. *Metody Podzemnoy Razrabotki Mestorozhdeniy*—Methods for Underground Working of Mineral Occurrences. Metallurgizdat, 1948) and the unified classification of methods and systems for working deposits of solid minerals compiled by L. Shevyakov (L. Shevyakov. *Razrabotka Mestorozhdeniy Poleznykh Iskopayemykh*—The Working of Mineral Deposits. Ugletekhizdat, 1951). These classifications are based on various features. The classification proposed by L. Shevyakov is based on two main characteristics: the disposition of development workings in bedded deposits and control of enclosing country rocks in unstratified deposits. Both features are very characteristic of such deposits. The author believes that in the future unified classifications a single typical feature should be established for all deposits of solid minerals. The setting up of a unified classification would in no way prejudice the practical importance of those employed in individual branches of the industry, which will undoubtedly be used in the future in selecting mining methods and studying them in detail. G. Tsulukidze's classification has found no practical application since it is complex and unwieldy.

In studying individual mining methods we shall use the system designations following from the above-listed classification of methods for the working of ore deposits. Along with the full designation, we shall give brief (somewhat conventional) designations which, though not fully illustrative of the system, have found wide application in the practice of mining ore deposits and in the mining literature. Individual mining methods are described in the following order: the mining, technical and geological conditions and practical applications of the systems; the design features of the method, including the general sequence of extraction, development work, and stoping operations; description of the currently used variants of the method, including mining operations (each operation is discussed in more detail together with one of the system variants); technical and economic characteristics of the method.

General assessment of the methods and their final selection are discussed in the concluding section dealing with the mining methods.

5-2. OPEN-STOPE METHODS OF MINING (CLASS I)

General

In the mining methods of class I the stope area is not artificially supported or filled during extraction, but remains open and supported by permanent or temporary natural ore pillars or, less frequently, by artificial protective stumps. Timbering employed in some methods is of temporary or auxiliary nature.

After the entire ore body or part thereof has been extracted the worked-out space with the abandoned permanent natural pillars continues to remain open. To achieve more complete extraction of the pillars containing valuable ore the enclosing country rock is artificially caved or the mined-out space filled with waste.

The main condition for the use of the open-stope methods of mining is considerable stability of the ore and the host rocks which would allow large areas of exposure to be left.

Open-stope methods of mining are effective and diverse. They are widely used in the mining of Soviet ore deposits, their share in the mining of iron and non-ferrous metal ores comprising about 25 and 30 per cent, respectively. A series of effective technological measures carried out in recent years at the mines of the USSR to improve and make the most of the open-stope mining methods (introduction of sectional drill steel and long blast holes[1], highly

[1] Hereinafter by a borehole is meant an artificial cylindrical depression in the rock measuring over 75 mm in diameter with a depth of up to 5 m, or of any diameter with a depth exceeding 5 m. By a hole is meant a similar depression with a diameter less than 75 mm and a depth less than 5 m (see classification

efficient equipment for loading and delivery of mined muck, optimal design standards for the various mining methods) give us grounds to expect an increase in the share of these methods in the mining of ore occurrences.

The drawback of some methods of this class is a considerable loss of the ore, reaching 15-20 per cent and even more (in the mining of salt deposits these losses are as high as 40-50 per cent).

Class I includes the following methods:

1. *Methods of block mining on strike*:

overhand stoping with stull timbering;

with stull timbering and partial or slice shrinkage of ore on stages;

underhand stoping of ore;

with breaking down of the vein matter from subdrifts (the two last-named methods are employed as an exception).

2. *Room mining*:

with sublevel breaking of ore from drifts or cross-drifts; with level-wise breaking by means of deep boreholes.

3. *Breast-stoping of ore*:

with or without stepped stopes.

4. *Room-and-pillar methods of mining*:

underhand stoping with breaking of ore by means of blast-holes;

overhand stoping with breaking of ore by means of blast-holes;

with breaking of ore by means of long holes over the entire area or throughout the height of the room.

Below we describe the most typical mining methods of class I.

Overhand Stoping with Stull Timbering

This system was formerly widely used in mining lode deposits, especially those of gold and rare metals (at present it has limited application).

The conditions for its use are: stable ore and host rocks; dip angle predominantly in excess of 45-50°; ore-body thickness ranging from 0.7-1 to 2-3 m (on rare occasions 4-5 m).

With this method of mining (Fig. 142) the ore body is worked out level-wise, the average level interval being 30-50 m. In the field a level interval height of up to 70 m has been adopted on some occasions, but this height is not typical and cannot be recommended for deposits with a varying mode of occurrence.

Each level is divided along the strike into blocks 40-50 m long (in individual cases up to 80-100 m long). With ore and host rocks

and terminology for mining machines, Academy of Sciences of the USSR). In conformity with the term accepted in mining practice blast boreholes longer than 12-15 m are called long, or deep holes. Boreholes drilled with an air-hammer by means of sectional-drill steel are called sectional-drill-steel holes.

of considerable stability and an ore body of limited thickness no pillars are, as a rule, left near the raises. Sill and arch pillars are left in extracting ores of little value and with an ore body more than 2 m thick. When leaving sill pillars short ore chutes are cut from the drift every 5-6 m. In mining valuable ores and with an ore body less than 2 m thick no sill or arch pillars are left. In such cases strut timbers are set up in the roof of the drift with a flooring of thick poles or round lagging laid thereupon and draw holes arranged

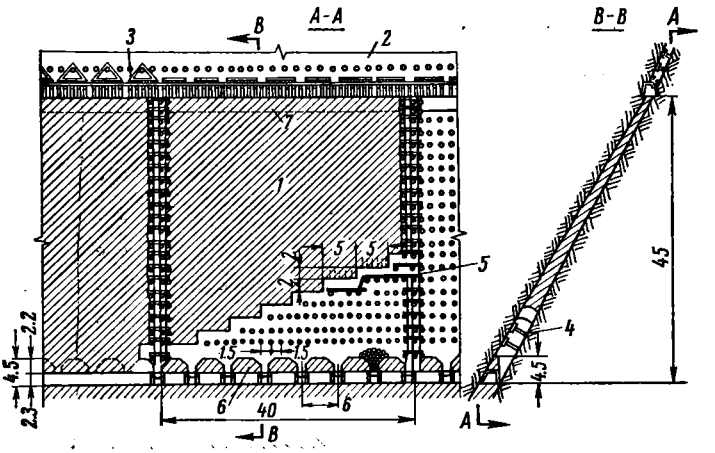


Fig. 142. Extraction of a block by overhand method with stull timbering (the production floor is worked with abandonment of sill pillars; the top level is mined through ore-chutes arranged above the drift):
1—block; *2*—top level; *3*—ore-chute; *4*—timber post; *5*—stage; *6*—sill pillar; *7*—arch pillar

every 5-6 m. Blocks in the level may be worked from the shaft towards the mine-field boundary and vice versa or, when necessary, simultaneously along the entire length of the level.

Development openings in the block include the upper and lower drifts and also two or three compartment raises.

After the excavation of level drifts and raises, stoping is started to one or both sides of the raise. The general advance of stoping in the block is from bottom up and of each bench, along the strike.

The ore is drawn in a series of horizontal slices, each lower slice being extracted in advance of the upper one; this makes possible overhand stoping, which is characteristic of this mining method. The advance rate of the underlying slice is taken within the range from 4 to 10-12 m. Stoping is commenced with the driftage of an advance heading (when a sill pillar is left), or with the extraction of the first slice in the back of the drift (in mining without leaving sill pillars). The ore in stopes is broken down by firing blast-holes

up to 2-2.5 m deep, drilled with jack hammers, or more often with hammer drill stopers. In the system under consideration extensive use of multiface and multihammer methods can be made. The ore broken in overhand stopes runs by gravity down the stope area along ground or timbered ore chutes towards pull holes.

Strut timbering is used for laying down stages and for the passage of workers through the stoped-out area. It is maintained either in production slices only, or throughout the entire worked-out area of the block. In the former case the broken ore is thrown down into the stope area (Fig. 143*a*), in the latter case into extensible ore chutes

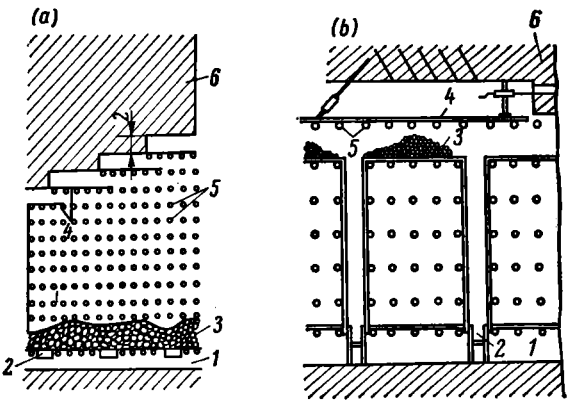


Fig. 143. Variants of overhand stoping:
a—with ore-chutes; *b*—without ore-chutes; *1*—drift; *2*—chute; *3*—broken ore; *4*—flooring; *5*—stull timbering posts; *6*—solid ore in place

(these are made by boarding up strut timbering) (Fig. 143*b*). The spacing between struts (on strike and dip) depends upon the stability of the enclosing country rocks. In the majority of cases the distance between strut rows is taken to equal the bench height, i.e. 1.8-2.2 m, while struts in individual rows are spaced at 1-1.5-2 m.

In the method under consideration strut timbering usually serves auxiliary purposes. The setting up of simple strut timbering made up of props 150-200 mm in diameter in a steeply dipping ore body is illustrated in Fig. 139. A strut at the foot wall is placed into a hitch 3-5 cm deep in strong rock and 15-20 cm deep in soft rock. At the hanging wall the strut is thrust against a headpiece—a sawn-off end of a board 5-10 cm thick. Where necessary a wedge is driven between the headpiece and the rock. The strut is not put up normally to the walls, but at an angle of 3-5° to the normal. This deviation prevents the struts from shifting downwards under the effect of blows from above, of the pressure exerted by the ore and mine-fill

lying on the timber and also of slight displacements of the hanging wall.

Overhand stoping with strut timbering enables the ore to be sorted on stages right in the stope area. In this case the discarded gangue is thrown into the space between the ore chutes.

Active stopes are ventilated by an air current flowing in from the haulageway and out to the upper horizon after sweeping the stopes.

The following basic safety rules should be observed during stoping operations: the back of the benches should be picked down thoroughly and in good time; the rate of advance of the underlying bench with respect to the overlying one should be at least 4 m (depending upon the operational conditions); the stages should be kept in good shape; ladderways should be provided between the benches; timberers must work with safety belts on.

The advantages of the method: the diversity of conditions for its application and the possibility of an easy switch-over to other systems (shrinkage stoping, mining with backfill or with reinforced support) in case of changes in the mining and geological conditions; the possibility of leaving barren rock inclusions (intrablock pillars); the feasibility of sorting the ore in the stopes, leaving the gangue in the goaf; high intensity of stoping operations (possibility of deployment of stoping in a large number of blocks); easy ventilation of active stopes; high coefficient of recovery; possibility of extraction of branchings, streaks and ore inclusions in the walls of the ore body.

The drawbacks of the method: high timber consumption for support and floorings; considerable dilution of the ore in mining narrow ore bodies (this may be largely eliminated by selective mining and sorting of the ore in stopes).

Measures to increase the effectiveness of the method may include multistope and multimachine drilling with extensive utilization of stopers and increased length of benches to ensure the necessary work front for the driller throughout the shift; reduction of the shovelling distance by using ancillary inclined slopes for ore movement to ore chutes; the application of movable metal support with a view to cutting down expenses on timbering; adoption of high benches when no picking of ore is necessary; this enables the driller's work front to be increased and the haulage conditions improved.

Among other steps mention can be made of special combination drilling-and-conveying units and bulk blasting of the drilled holes. These measures have been proposed and carried out in the field by the Moscow Mining Institute under the guidance of Prof. R. Kaplunov. The ore is extracted starting from the level boundary in a continuous vertical face without dividing the level into benches and setting up stull timbering. The carriage of a combination drill-

ing-and-conveying unit is suspended by a rope and moved by a special hoist installed in the upper drift. The drillers make holes with air drifters mounted on the suspended carriage. This unit was employed in tungsten and gold mines and made it possible to increase the level interval height, reduce the width of the stope down to 0.7-0.9 m and raise the driller's efficiency, this being particularly important in working thin and steep ore bodies (expenses are cut down and the quality of broken ore improved).

The technical and economic characteristics of overhand stoping with stull timbering are listed in a combined table of the characteristics of open-stope mining methods.

Mining with Stull Timbering and Skeleton Shrinkage of Ore on Stages

This system is successfully employed in the working of deposits of non-ferrous metal ores.

The method involving partial shrinkage of ore is applied in conditions similar to those where overhand stoping is used, but when

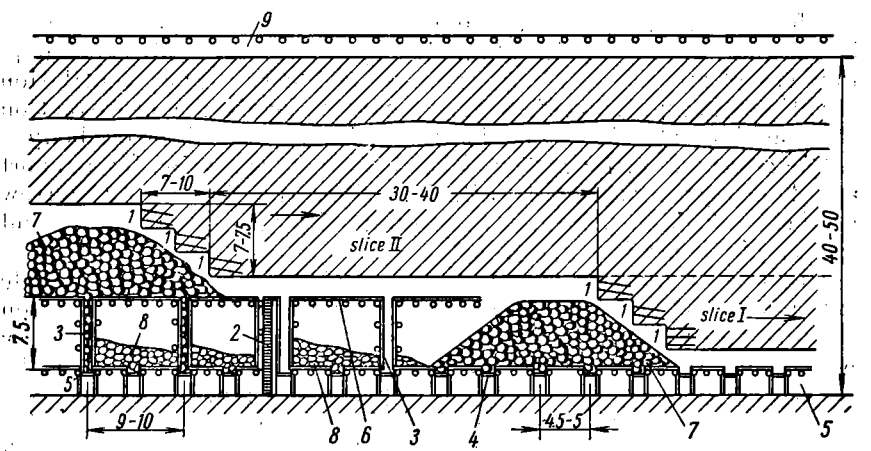


Fig. 144. Overhand mining with stull timbering and skeleton shrinkage of ore: *1*—working benches; *2*—ore-chute with ladderway; *3*—ore-chute; *4*—loading chute; *5*—haulage drift; *6*—flooring; *7*—broken ore; *8*—barren rock; *9*—ventilating drift; arrow shows direction of slicing

the width of the ore body does not exceed 2.5-3 m. A distinctive feature of the method is the extraction of the ore in slices up to 7-7.5 m high and temporary abandonment of the broken ore on a stage of the stull timbering (Fig. 144). The broken ore serves as a flooring for the workers engaged in drilling a three-bench stope

with jack-hammers or air stopers. Drilling is accompanied by the discharge of the stored ore (along the back slope) from which gangue is picked out and placed between ore chutes. 65% of the broken ore is handled through the back and 35% through the front slopes. Partial removal of the ore from the front slope is explained by the need to make free room near the benches. Individual slices are extracted with a lag of 30-40 m.

The advantages of the method: substantial reduction in timber consumption and in the scope of timbering, since stulls and floorings are spaced at 7-7.5 m instead of 2-2.5 m; elimination of occasional knocking down of stull props and damage to the flooring during blasting; drilling, timbering and ore cleaning operations can be synchronized, this increasing the intensity of stoping; stocks of broken ore are available, which ensures its uniform delivery.

Comparing the method under consideration with that involving full shrinkage of the ore we should note the following merits of the former. It can be used in more complicated mining conditions—with ores liable to packing, oxidation and self-ignition, also when sorting is necessary and frequent changes occur in the mode of occurrence of the ore body. These conditions exclude the application of the system with full shrinkage of ore throughout the entire level interval.

In order to enhance the effectiveness of the method involving partial shrinkage of the ore use is made of portable drawing chutes through which the ore is pulled and also its breaking and shrinkage in horizontal slices along the entire length of the block.

Variant of the Method of Mining with Stull Timbering and Skeleton (Slice-wise) Shrinkage of Ore. This variant (Fig. 145) consists essentially in the following. A block 50-60 m long and 40-60 m high is extracted in separate strips from bottom up, each of the strips, 4-6 m high, being worked out in separate horizontal slices 1.3-1.4 m high and the broken ore stored on a stage fixed to stull props along the entire length of the block. Each strip consists of three slices. After the extraction of the last slice the stored ore is drawn starting from the block boundaries through portable chutes in the flooring. To make such chutes, round lagging is rolled from one place to another by using hooks. The chutes are made in advance near the block boundaries when laying the flooring. As the chutes move on the ore discharge site moves too. The round lagging rests upon girders fastened to strut props.

The variant of the system with slice shrinkage of ore can be used under mining and geological conditions similar to those in which ordinary overhand stoping with stull timbering is employed, and also where complete shrinkage is unfeasible owing to the small width of the ore body, local flattening out, irregular mode of occur-

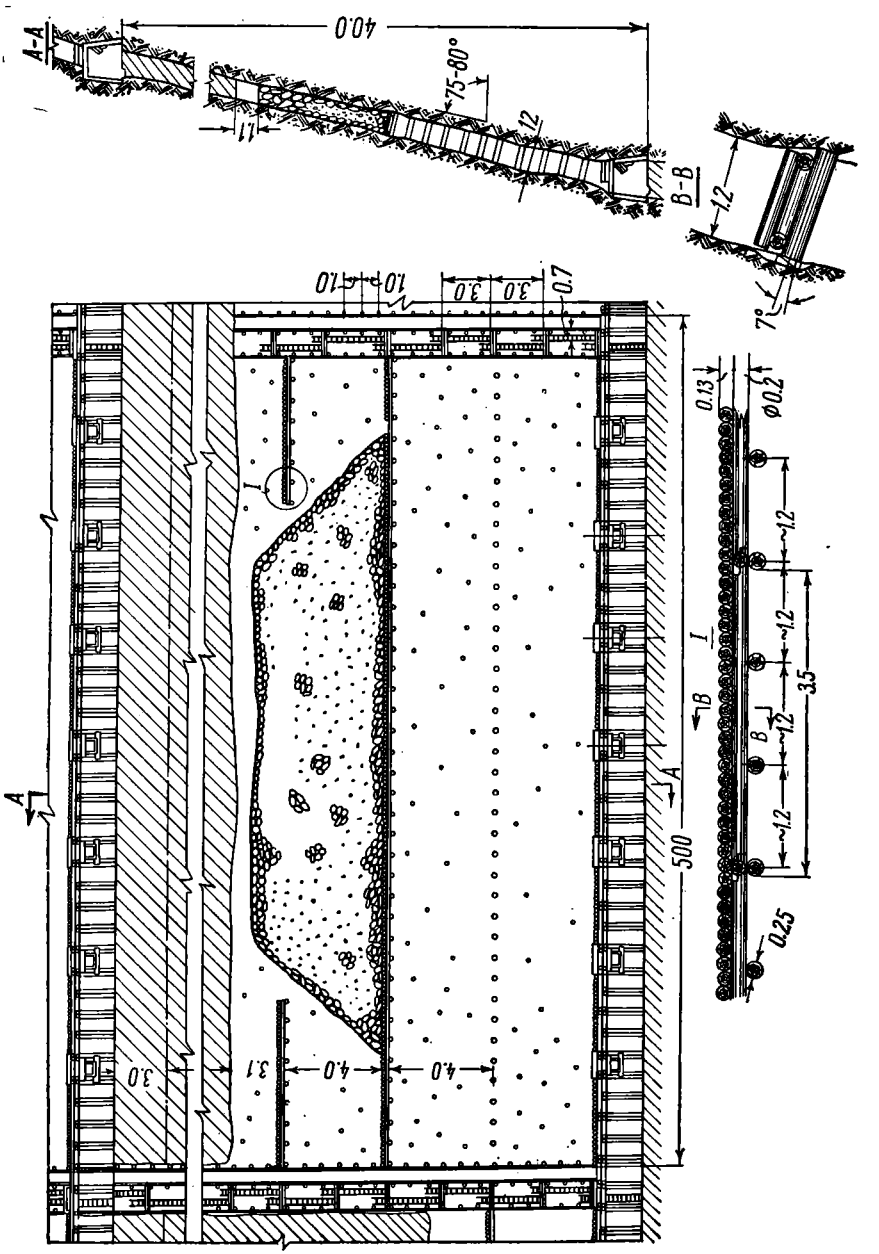


Fig. 145. Variant of the stull-floor method of mining with slicing and shrinkage of ore

rence, tendency of the ore to packing and oxidizing, and insufficient rigidity of the host rock. This variant is used in the field with lodes ranging from 0.5-0.7 to 2-2.5 m in width. Its major advantages as compared to the ordinary method of mining with stull timbering are: greater safety, increased efficiency per faceman, high intensity of extraction of the mineral and decreasing timber consumption by half. Its shortcoming is the complexity of ore sorting.

Block Mining on Strike with Extraction of the Vein Matter from Subdrifts (Sublevel Method of Mining Extremely Thin Veins)

Owing to the specific nature of extracting the vein matter by comparison with usual working through subdrifts we find it expedient to consider this system as an independent one.

Subdrift mining of very thin veins (Fig. 146) was proposed by Engineer V. Mertsalov before World War II. The system has been

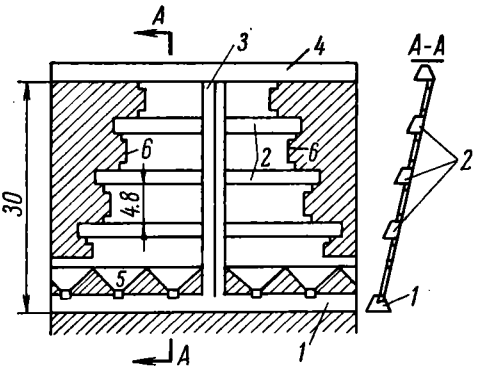


Fig. 146. Mining by blocks on strike with breaking the vein matter from subdrifts:
1—haulage drift; *2*—subdrift; *3*—raise; *4*—ventilating drift; *5*—funnel-shaped draw hole; *6*—bench

successfully used at Soviet gold, tin and nickel mines in working very thin veins (from 8-10 to 20-30 cm thick) with stable enclosing rocks having regular contacts and dip angles of 75-80°.

In this method a horizon 30-40 m high is divided by raises into a series of blocks 50-70 m long. Funnel-shaped ore-receiving holes with chutes spaced at 7 m are made over the haulageway. Sublevel drifts 2 × 0.8 m in cross-section are driven on both sides of the raise as the stopes in the block advance with a lead of 2.5-3 m. The sublevel interval height is 6.5-7.5 m. When stoping, up-and-down holes are drilled from subdrifts along the vein every 0.4 m. As a result of hole firing a narrow slit, somewhat wider than the vein, is formed. The broken ore slides along it down into the receiving funnels.

The chief advantages of the method are: higher efficiency per faceman in producing the vein matter, greater intensity of stoping and lower (by 25-30%) cost per 1 cu m of the extracted vein matter "free in the drift" as against the corresponding figures for the stull-set method of mining. The system described finds little application because of the rigid conditions it must meet.

Room Mining with Sublevel Breaking of Ore From Drifts or Crossdrifts (Sublevel Method of Mining)

The method is widely used in working Soviet ore deposits. Its proportional share in the total output of the Krivoy Rog iron-ore basin reaches about 10%. It is often employed in other iron-ore regions as well.

In mining by sublevel drifts or crossdrifts the horizon is divided into rooms and pillars. The rooms are extracted in the first and the pillars in the second working. The ore in the rooms is extracted either along or across the strike, depending upon the thickness of the ore body. In the former case the ore in the rooms is broken from subdrifts, and in the latter from sublevel crossdrifts.

Sublevel stoping (Fig. 147). The basic conditions for the use of the method are: considerable stability of the ore and host rocks; dip angle of the ore body not less than 50°; thickness of the ore body from 5-6 to 20-25 m (when it exceeds 20-25 m the rooms are extracted across the strike).

The ore body is worked by horizons with an interval height of 50 to 75-100 m. With strong ore and very stable enclosing rocks it is possible and expedient to increase the level interval to 70 m and more.

The horizon is broken up into blocks 50-60 m long each. With a high level interval the length of a block often reaches 100-120 m. The length of a block should be in line with permissible exposure areas in the roof and walls of the room. With wide rooms the length of the block is smaller. In the Krivoy Rog basin, where the ore strength amounts to $f = 8\text{-}12$ and the enclosing rocks are very stable, exposure areas in the hanging wall reach 2,000-3,000 sq m. Each block is divided into a room and a pillar. The width of the pillar is at least 6 m, the thickness of the floor pillar being 0.4-0.6 of the room width. The overall thickness of the floor (from the haulage elevation mark down to that of undercutting) varies from 10 to 15 m with a grizzly level and 6 to 10 m with a scraper level.

Stoping in the room may be done in one direction or in two directions—from the centre towards the boundaries. In the latter case the shortcomings due to the large length of the room (for instance difficulties in delivering supplies to the stope) are eliminated and its yield is increased.

Major and subsidiary development. In this method the major development workings in the block include level haulage (*1*) and vent (*11*) drifts, and raises (*2*) (Fig. 147) cut in intervening pillars.

The subsidiary development openings include: subdrifts (*6*) (with the short-hole method these are driven along the room axis); a set of workings associated with grizzly or scraper levels and with

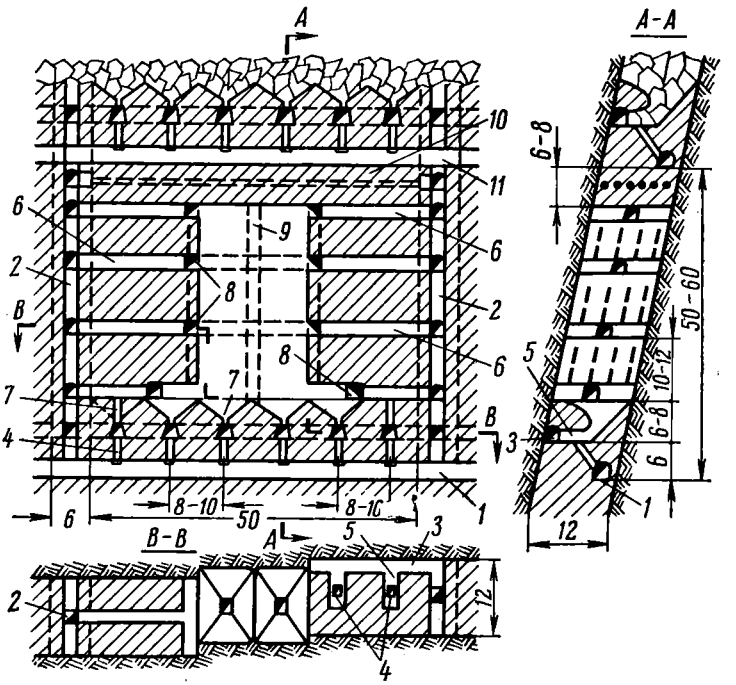


Fig. 147. Typical variant of the sublevel method of mining with a grizzly level:
1—haulage drift; *2*—raise; *3*—grizzly drift; *4*—ore-chute; *5*—dozing chamber; *6*—subdrift; *7*—finger raise; *8*—open sublevel crossdrift; *9*—cut-out raise; *10*—floor pillar; *11*—ventilating drift

the discharge of ore onto the haulageway; a cut-out raise (*9*), which is made near one of the intervening pillars or in the centre of the room. The distance between subdrifts corresponds to the adopted sublevel interval, which varies within a very broad range. The method of ore breaking is of decisive importance in selecting the sublevel interval. With shallow blast-holes it is taken to be up to 6 m, with sectional-steel air-hammer drilling 10-12 m and with longholing up to 20-25 m.

The workings linked with the grizzly level and with the pulling of ore onto the haulage horizon include: a grizzly drift (*3*), dozing

chambers (*5*), finger raises and draw holes (*7*). Each dozing chamber supplies one draw hole (more detailed information on ore crushing and the design of grizzly chambers will be given below).

The dozing chamber communicates with the haulage drift through a short ore chute (*4*).

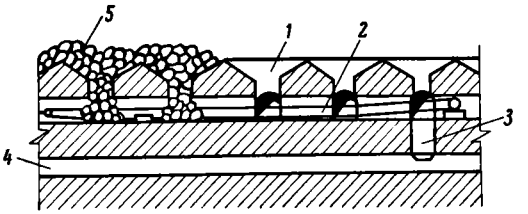


Fig. 148. Workings in the bottom portion of a room excavated along the strike, with a scraper level (side elevation on strike):
1—drawhole; *2*—scraper drift; *3*—ore-chute; *4*—haulage drift; *5*—broken ore

Prior to the excavation of funnel-shaped draw holes the ore body is undercut by crossdrifts (*8*) at the level of the bottom subdrift.

In the last 15 years at many mining enterprises of this country

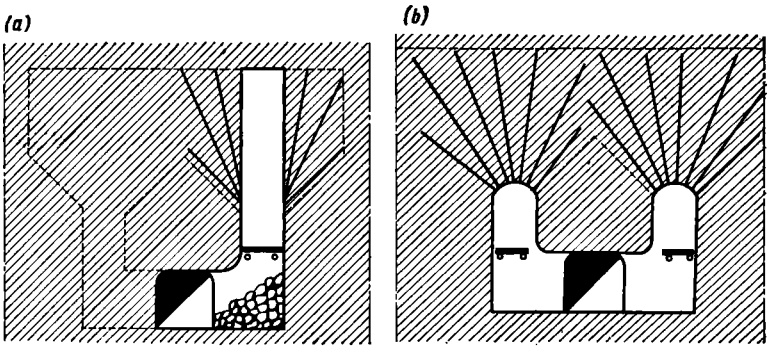


Fig. 149. Undercutting by a round of sectional-steel drill holes

the grizzly horizon has been replaced by the scraper level (Fig. 148). Its use greatly reduces the number of ore chutes and eliminates the excavation of dozing chambers and associated workings. This replacement, however, is feasible only when taking ores of limited strength which do not require much secondary breaking, or when changing over to drawing large-sized ore.

Now we will briefly describe funnel-wise undercutting applied in the method under review and in other methods of working ore deposits.

In undercutting, ore-receiving funnels are lined up in one or two rows, depending upon the thickness of the ore body. If it is up to 10-12 m the funnels form a single row, with thicker ore bodies they are arranged in two rows. Funnels are formed by raising ore-passes and their subsequent enlargement. In a strong ore the ore-pass raises are cut throughout the height of funnels (Fig. 149 *a*), or to a lesser

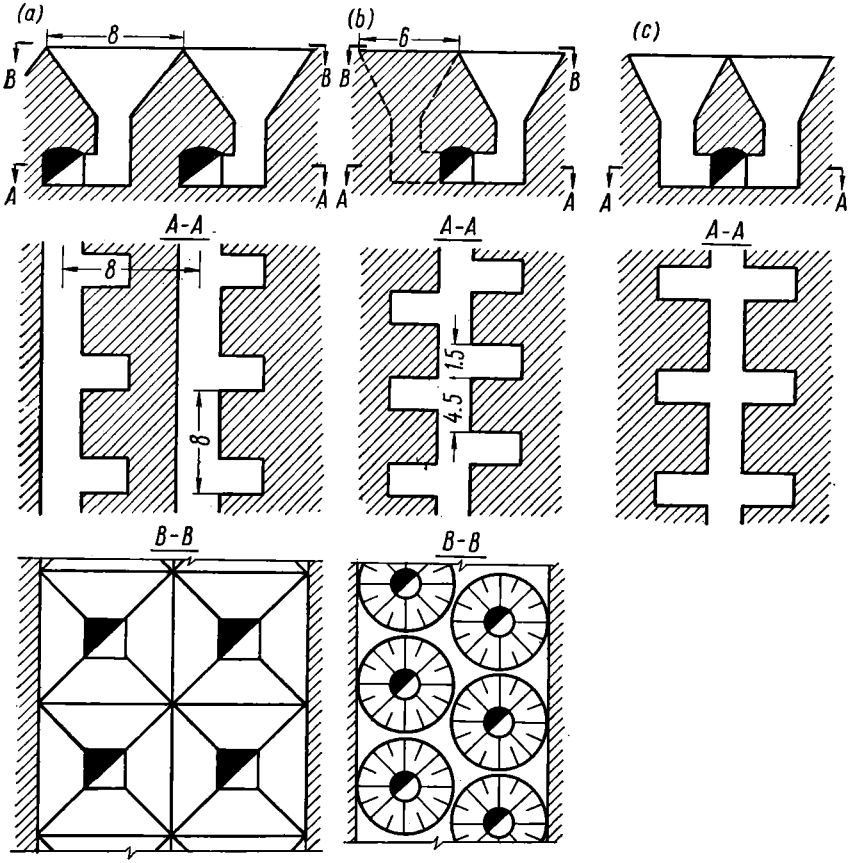


Fig. 150. Disposition of finger raises

height (Fig. 149*b*) in an ore of medium strength, with subsequent sectional-steel drilling and firing of holes within the funnel. With funnels lined up in two rows they may be placed so as to ensure uni- or bilateral drawing of ore (Fig. 150*a*). With bilateral drawing of ore the pull-holes may be sited symmetrically or in a staggered order (Fig. 150*b*, *c*). In arranging grizzly or scraper levels it is extre-

mely important to provide a proper junction between the bottom part of the ore-pass and the workings of the corresponding horizon, since the ore must be drawn freely with strict observance of the safety rules.

Figure 151 illustrates the junction of a finger raise and a scraper working. Finger raises are made from a horizontal cutout and are sited at a distance from the scram working such that the ore being pulled fills not more than half, or at the most $^2/_3$ of the scram working width, and the overhanging ore is held securely in place. Scram workings have a width double that of the scraper. The finger raise

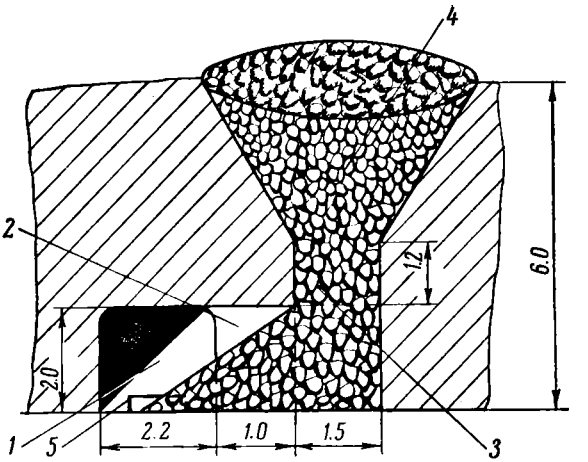


Fig. 151. The junction of a finger raise and a scram working:
1—scram working; *2*—horizontal cutout; *3*—finger raise; *4*—funnel; *5*—scraper

section measures 2-4 sq m, depending upon the size of the ore to be drawn. The hole diameter should not be less than 3-4 times the size of the biggest chunk of the ore pulled.

Figure 151 indicates approximate dimensions of the corresponding workings. Labour efficiency in funnel undercutting with sectional-steel drilling is 40-70 tons per shift.

The above-described method of undercutting with the use of sectional-steel drill holes is more effective than that with funnel undercutting and shallow blast-holes. Among the other effective methods of undercutting employed in recent years trench undercutting and undercutting with flat room floors (Figs. 152, 153) are worthy of notice.

The trench method of undercutting allows trenches to be cut by longholing, the construction of the room floor is simplified and the amount of labour-consuming subsidiary development work redu-

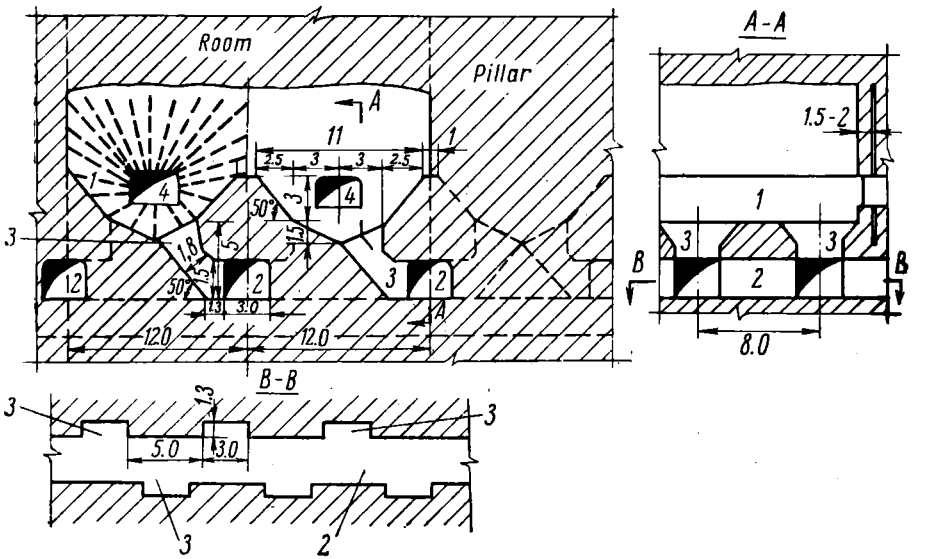


Fig. 152. Scraper haulage in a room with a trenched floor:
1—trench; *2*—scram working; *3*—finger raise; *4*—opening from which deep holes are drilled to make the trench

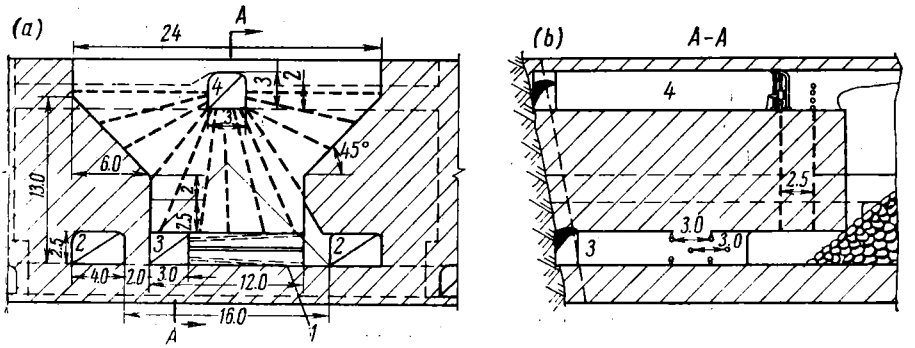


Fig. 153. Undercutting a room with a flat floor:
1—flat floor of the room; *2*—scraper working; *3-4*—openings from which deep holes are drilled in undercutting the room

ced. In this method the yield per driller per shift is 60-100 tons. The construction of the flat floor of the room is still simpler. By a flat floor is meant a floor without ore-passes or funnels and where the haulage, drawing-off and floor levels are combined into one. In this method of undercutting long blast-holes are widely used. Long holes are drilled from special openings, as shown in Fig. 153, or from small rooms excavated from scraper workings (at an angle, from bottom up). Flat-floor rooms with slushing of the discharged ore have been successfully used in the USSR, they raised the efficiency per faceman for the system as a whole by as much as 25-30% and cut down the cost of ore by 8-11%.

A still greater effect may be achieved by making flat floors with a vibratory-discharge chute in order to draw the ore directly to the haulage horizon without slushing it (Fig. 154). Vibratory discharge (pulling of ore with the aid of vibration) simplifies the design of the block floors. With such floors the volume of workings drops from 2.5-5 cu m to 1-1.5 cu m per sq m of the block area (the draw level is combined with the haulage level). Large-scale tests of various vibratory installations in the USSR demonstrated their efficiency to be from 220 to 290 tons/hr. Their other specifications are: (1) angle of inclination of the discharge trough from 7-10 to 17-23°; (2) amplitude of vibrations of the operating mechanism from 0.6-0.9 to 2-3 mm; (3) vibration frequency from 1,000 to 2,800 per minute; (4) electric motor capacity 14-28 kW; (5) overall dimensions: length up to 6,000 mm, width up to 1,700 mm, height from 500 to 2,800 mm. The size of chunks up to 1,000 mm. Many of the vibro-unit models can be operated under amassed bulk of ore.

The application of vibro-discharge greatly reduces expenditures on the drawing, delivery and loading of ore and increases the efficiency of ore recovery in the block 2-3-fold as compared to scraper delivery. The technological potentialities of vibro-discharge are far from exhausted and constitute only 25-50% of the rated capacity. The main task in increasing the efficiency of vibro-discharge equipment is to ensure quick spotting of mine-cars for loading and supplying larger tonnages of ore to these units (pulling of ore from 2 funnels through one draw-hole, entrainment of the broken ore by a blast wave to a restricted number of draw-holes, manufacturing of light vibro-units which can be easily shifted to different discharge points).

Stoping in sublevel mining (Fig. 147) is started by cutting the room off from the pillar and enlarging the cut-out raise, thus forming a so-called opening slot 2-3 m wide.

The raise can be widened to form an opening slot by using an underhand stope (starting from the upper sublevel), breaking the ore by firing shallow blast-holes, or by driving small annular pockets (2) from the cut-out raise (1) at each sublevel with subsequent

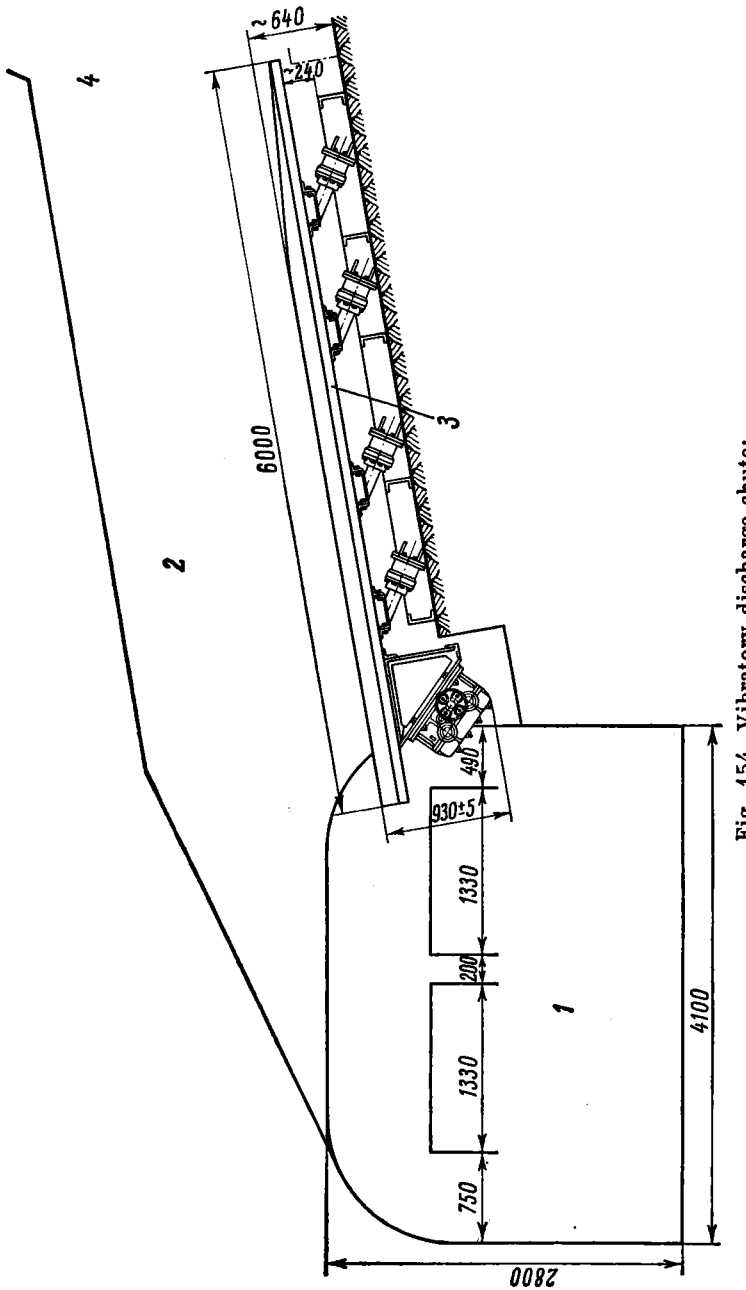


Fig. 154. Vibratory discharge chute:

1—drift; *2*—inclined opening connecting the room floor with the drift; *3*—vibrofeeder; *4*—discharge outlet from the room

breaking of ore therefrom by firing sectional-steel drill holes (Fig. 155). The annular pockets are gradually enlarged from the subdrift (3) by drilling and exploding new rounds of sectional-steel drill holes. A consecutive series of such operations results in an opening slot of the desired size. This method of forming an opening slot is widely employed.

Lately a more effective method of excavating an opening slot has been used involving drilling of long holes and breaking down

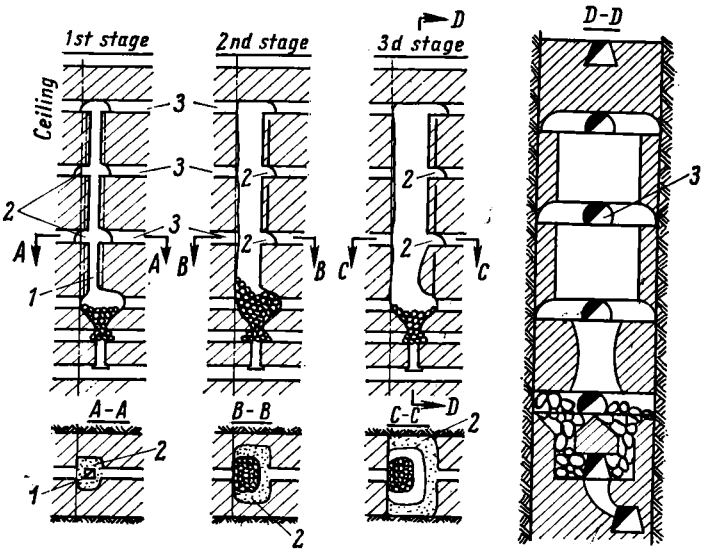


Fig. 155. The formation of an opening slot by longholing:
1—cut-out raise; *2*—pocket; *3*—sublevel drift

the ore into the raise. The long holes are drilled parallel to the raise. The latter may be completed in a short time by using a set of drilling equipment with a movable stage (the peak rate of excavating raises with such a set so far achieved is 257 m/month). The formation of an opening slot is followed by undercutting the block over a corresponding length, excavation of funnels or a trench and, finally, by stoping operations.

In sublevel mining the general front line of the stope can be run vertically or stepwise. A stepwise stope front line is used in breaking ore in the sublevel by means of short holes. This method is unproductive and rarely used. At present ore in sublevels is commonly taken by using blast-holes. These can be drilled parallel to one another from bottom up (self-cleaning holes), thus creating optimal conditions for the removal of dust from the holes. This arrangement

of blast-holes is shown in Fig. 147. In breaking ore by blast-holes the stope front is run vertically as a rule. Breaking of ore by firing sectional-steel drill holes is described below.

To break ore by a parallel round of blast-holes, its solid mass is drilled from open crossdrifts (pockets) over each sublevel drift. These crossdrifts are regularly pushed forward at the ends of sublevel drifts. Their height is equal to or somewhat greater than that of the subdrift, their width being 1.5-2.5 m. The drivage of an open pocket is followed by drilling vertical or inclined blast-holes 6-8 m deep with the use of hammer stopers and sectional steel. Blast-holes are spaced at 3-3.5 m and placed at a distance of 1 m from the contact. It should be borne in mind that as the spacing between the blast-holes increases, so does the yield of large-sized lumps, and this raises the cost of secondary crushing and brings down the draw. With sectional-steel drill holes the overall sublevel interval is set at 8-12 m. The drilling of the back of thepocket along its entire length is followed by the charging of blast-holes.

The drawback in the breaking of ore by using parallel sectional-steel drill holes is the large amount of work that has to be done in each sublevel when running open pockets. This disadvantage si eliminated by using a fan-shaped disposition of blast-holes drilled from a subdrift with heavy column-mounted drifters (Fig. 156).

With this method a fan-shaped round of blast-holes 10-12 m deep are drilled with column drifters weighing 70-80 kg each and mounted on upright jack columns. The drilling rig consists of a set of drill rods (screwed together by means of threaded sleeves) and drill bits 82-85 mm in diameter reinforced with a hard alloy. Blast-holes are flushed with water (wet drilling).

Sublevel mining by the methods of Zinkov (fan-shaped rounds of blast-holes) and Mitrofanov (multihammer drilling) made it possible to increase the output per driller per shift to 250 tons and raise the ore tonnage per 1 metre of the blast-hole to 25-40 tons (chiefly by augmenting burden to 2.5-3.0 m).

When assessing new methods of stoping operations it should be noted that with the old method of breaking ore from open pockets the average output per driller was equal to 70-140 tons per shift, the average ore yield per 1 metre of the blast-hole being 15-20 tons.

Progressive workers, technicians and engineers have made a number of suggestions for further improvement of stoping. One of the most effective among them was breaking of ore by longholing.

Underground stoping of ore by using long holes is one of the outstanding technological achievements of the Soviet mining industry. The advantages of longholing include: high yield of ore per 1 metre of blast-hole (up to 50-70 tons), better health and hygienic conditions of work for drillers, high efficiency and low cost of

stoping operations. Longholing was first introduced in the USSR at the mine "Pervomaiskaya" (Krivoy Rog basin) as far back as 1935. Today it is widely practised at a number of mines in the USSR and other countries in taking ore from rooms, blocks and pillars. With ores having a strength of up to f = 7-8 long holes can be drilled with small rotary rigs using hard-alloy-tipped bits, short column

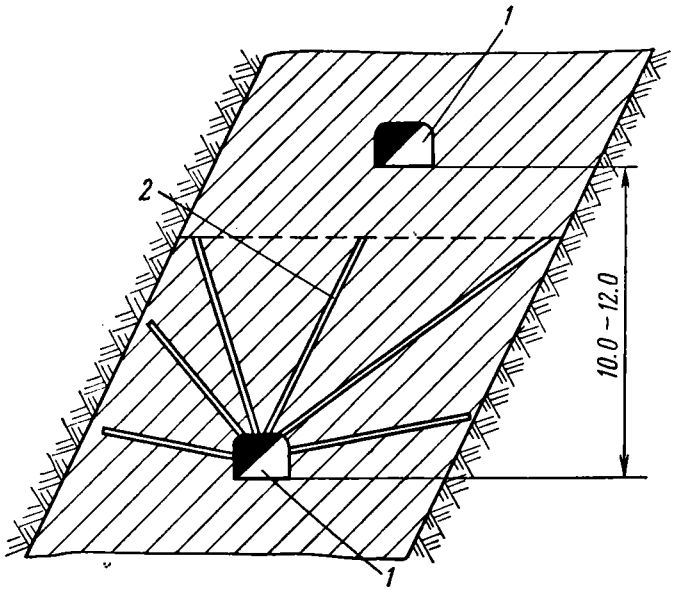


Fig. 156. Drilling over a sublevel with a fan-shaped round of long holes using Zinkov's method:
1—sublevel drift; *2*—long hole

pipes and extension rods. Small-diameter blast-holes (up to 75-85 mm) 50 m deep are drilled with specially designed units using water- or clay-solution flushing. The bit consists of a ring equipped with hard-alloy cutters. The drilling rate of blast-holes in grounds of limited strength reaches 20-25 m per shift, the average figure being 12-15 m per shift. With proper conduct of work deviation of deep holes does not exceed 1-2 m per 40 m. Blast-holes in a strip are run parallel or fan-wise.

In the last 15 years rotary rigs with rock-roller bits and drilling units with air striking pins (punchers) have found extensive application in the mines of the USSR. These units enable blast-holes to be drilled in hard ground.

An extremely ingenious design of a drilling unit (БА-100) has been worked out by the West-Siberian branch of the Academy of Sciences of the USSR. It served as a prototype for other drilling units.

The БА-100 drilling unit is intended for drilling blast-holes 100 mm in diameter and 50 m deep in any direction. The drilling rate of the machine in medium-hard grounds amounts to 10-14 m/shift. The working part of the unit is an air striking pin

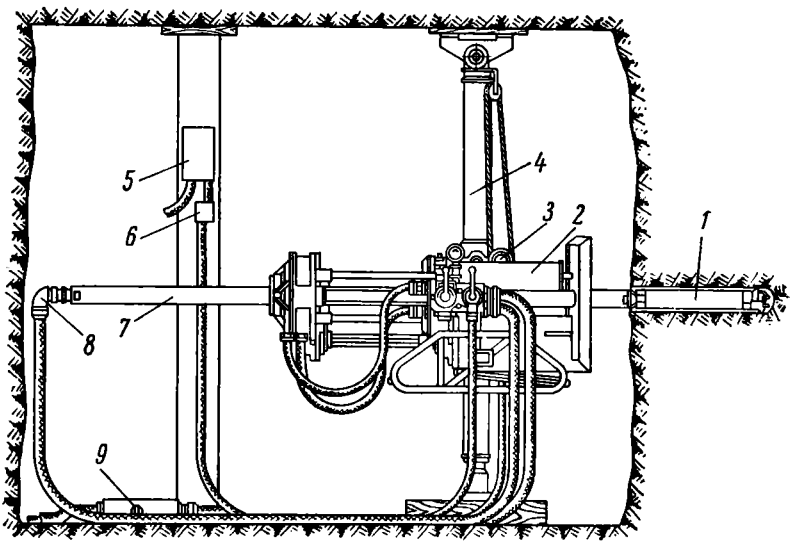


Fig. 157. HKP-100 drilling unit:
1—air striking pin; *2*—drill feed; *3*—hoisting winch; *4*—anchor post; *5*—starter; *6*—push-button controls; *7*—drill rod; *8*—stuffing box; *9*—automatic oiler

(puncher) which enters the blast-hole; it has no rotational mechanism. The rods that push the air striking pin into the blast-hole are rotated by a special electric motor. Compressed air for the air striking pin and water to flush the blast-holes are delivered through the rods. Among the latest designs of air-puncher-type drilling machines mention should be made of the ЛПС-3 drilling unit provided with an external control desk serving two units simultaneously.

The general view of the HKP-100 drilling unit is presented in Fig. 157. It consists of a machine, an air striking pin and a string of rods with air and electric connections. The drilling machine is semi-automatic (mechanized clutching of chucks and automated alternate gripping), it is mounted on a jack column and can turn around in any plane.

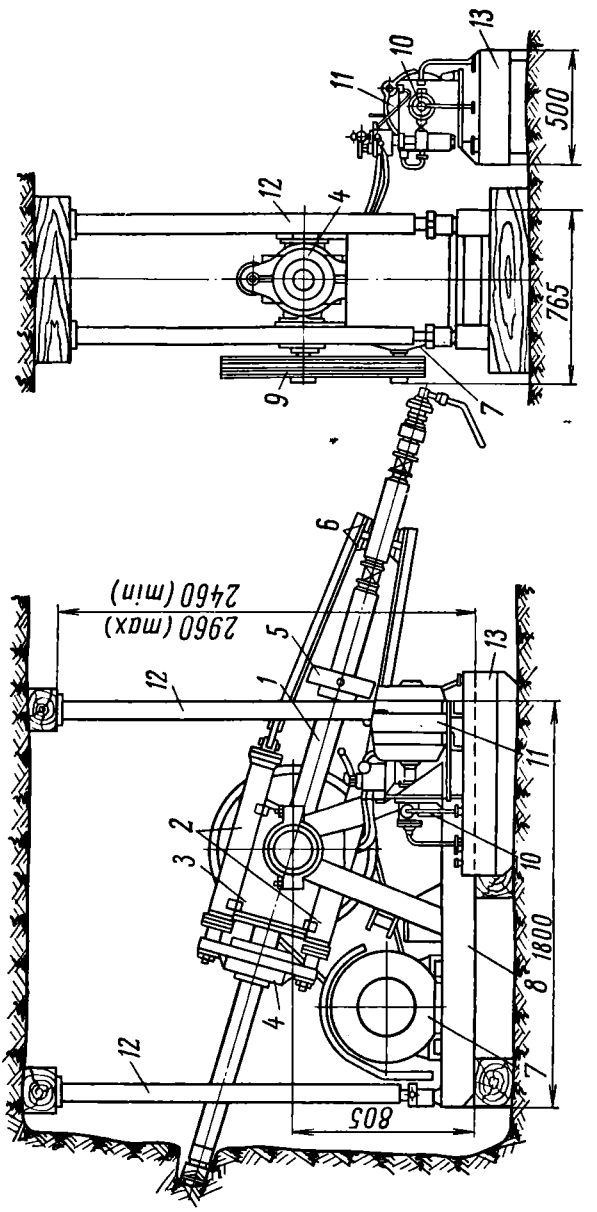


Fig. 159. СБ-4 drilling machine:
1—spindle; *2*—two hydraulic cylinders; *3*—rotation chuck; *4*—cross-piece; *5*—drill chuck; *6*—drill-steel holder; *7*—electric drive motor; *8*—frame; *9*—texrope transmission; *10*—pump; *11*—pump drive motor; *12*—jack columns; *13*—oil tank

diameter blast-holes are usually drilled with thin-faced bits. Full-hole drilling is used which permits the idle time of the machines to be cut down considerably (there is no need to withdraw the core). The life of a diamond-set bit fluctuates between 15 and 40 m, depending upon the hardness of the ground. The average drilling rate of a diamond-drilling rig is 10-30 m/shift, depending on the hardness of the ground, the diameter of the blast-hole and the drilling unit design. The yield of ore per 1 metre of blast-hole amounts to 15-20 tons. The advantages of breaking ore with the aid of diamond drill-holes include small yield of oversize and the possibility of drilling long holes in different directions and at a high rate of advance. At present new diamond drilling machines are being developed in the USSR and large-scale trials are underway at a number of mines.

Calculation of charges and firing of long holes. Many formulas for calculating blast-hole charges have been suggested. In the field the size of a charge in a blast-hole is usually estimated from explosives consumption per 1 cu m of the blasted bulk. On this basis the following formula for estimating the required diameter of a hole has been proposed:

$$d = \frac{w}{\sqrt{\frac{7.85\Delta\tau}{Km}}}$$

where d = diameter of the blast-hole, m;

w = burden (the line of the least resistance, or the shortest distance from the centre of the charge to the exposed surface), m;

Δ = explosives density in the charge (determined by consulting tables, depending upon the explosive used);

τ = relative length of the charge in the blast-hole, characterized by the degree of filling of the blast-hole with explosives; the adopted values are:

with a blast	hole 5 m	deep	0.7
	ditto 10	ditto	0.8
	ditto 30	ditto	0.9
	ditto 50	ditto	0.95

K = consumption of explosives, kg/m^3;

m = factor allowing for lamination of the rocks; it is taken to equal 0.7 to 1.3.

When the value of d is given, w can be determined.

Data on the average consumption of explosives (ammonite No.2) in the USSR in relation to the hardness of the ore are listed below. Explosives consumption figures are determined more accurately in each individual case from the data of experimental blasts with special reference to the particular conditions of blasting.

Ores of medium hardness: consumption in primary breaking 0.1-0.15 kg/ton, in secondary breaking 0.05-0.1 kg/ton.

Hard ores: consumption in primary breaking 0.2-0.25 kg/ton, in secondary breaking 0.1-0.2 kg/ton.

Very hard ores: consumption in primary breaking 0.30-0.45 kg/ton, in secondary breaking 0.3-0.4 kg/ton.

Among the explosives now employed in underground mining the wide application of a new explosive, igdanite, is worthy of note. This is a very simple explosive which is prepared directly at the workplace and consists of granulated ammoniacal saltpetre impregnated with a small amount (5-6%) of liquid fuel (solar oil or diesel fuel). As compared to ammonite (dynammon) igdanite is less sensitive to external effects and therefore can be used for mechanized charging. Priming of igdanite is done with an intermediate detonator made of ammonite or detonite. By employing pneumatic charging of blast-holes the charge density of igdanite may be brought up to 1.25 g/cm^3. Thus it can replace both ammonite and detonite and yield better results.

In accordance with the drilling equipment in use the diameter of deep holes is commonly adopted within the range from 75 to 150 m, with a burden of 3.5-5 m, blast-hole spacing of 3-4 m, blast-hole depth of 15-40 m, the yield of ore per 1 metre of blast-hole amounting to 30-70 tons. The blast-holes are run horizontally or vertically, parallel to one another, or fan-wise.

The explosives are inserted into blast-holes in long cartridges, using extensible stemmers or pneumatic charging devices. The blast-holes are fired either electrically or, more often, by means of a detonating cord (2 cord lines) with the use of short-delay detonators.

In recent years a new procedure of short-delay firing with the use of a pyrotechnical relay has found application in the mines of the Krivoy Rog basin. This method has been introduced at the mines by research workers of the Krivoy Rog Mining Institute (Prof. G. Malakhov, Associate Professor V. Lavrinenko, and others). The pyrotechnical relay allows the explosion to be delayed by 10, 20, 35 and 50 msec. By connecting relays in series it is possible to produce a delay of 200 msec, which fully meets mining requirements. One of the advantages of pyrotechnical relay blasting is a longer effect of the blast waves propagating in different directions on one and the same area of solid ore in situ (better fragmentation and smaller explosives consumption). It should be borne in mind that short-delay blasting ensures better integrity of pillars and the ceiling due to dispersed firing of charges. Multiple-shot firing of large charges should be done with the use of control and measuring instruments (linear detonator fuse bridges, voltmeters, etc.) according to a special plan of blasting operations with due regard for safety rules.

Drawing of ore in sublevel stoping. Ore broken in sublevels drops into receiving funnels from which it is pulled onto the scraper or grizzly level. In these horizons big ore lumps are crushed on special coarse screens made of rails, large-section steel beams or of wood-reinforced steel pipes, which have proved to be stronger than rails and beams. On the coarse screen the ore is commonly fragmented with the aid of explosives. In this case big lumps are first drilled with jackhammers. For ore fragmentation use is also made of slab charges, then the consumption of explosives for secondary breaking is greater, but secondary drilling is unnecessary. The ore broken into lumps measuring up to 300-400 mm (usually up to 300 mm) is drawn onto the haulage level, loaded into mine-cars and then delivered to the shaft. In the case of small-scale secondary breaking the chambers are arranged directly over the haulageway, and when the scope of this work is considerable, on a special grizzly level lying 4-8 m above the haulage horizon.

The grizzly level is connected with the haulage horizon through short ore-chutes. When laying out the grizzly level the area of one draw funnel should be kept within 70-80 m^2.

When working in grizzly chambers one must strictly observe the basic rule—to keep away from the discharge openings of the ore chutes. If some ore overhangs in an ore-chute big lumps are fragmented; to do this an explosive charge is brought to the site of the overhanging mass by means of a long pole. When firing the charge the miners should leave the grizzly chamber.

Due to the presence in mines of water supply trunk lines for wet drilling, ore overhangings in pull-holes are now effectively removed with water jets under a pressure of 2-3 atm.

The per-shift output of a miner on a grizzly level varies from 60 to 150 tons of ore, depending upon the conditions of its drawing. The additional consumption of explosives for secondary breaking is 0.05-0.40 kg/ton for all the ore pulled out, depending upon its hardness and the size of lumps. The layout of grizzly and secondary breaking chambers is shown in Fig. 160*a*.

A grizzly rests on steel beams (Fig. 160*b*). It consists of several rails (*1*) with turned up feet, joined together by bolts (*2*) and pipes (*3*). The grizzly has a slope of 1 : 15-1 : 20 and is laid down on steel beams (*4*).

With setting up of a scraper level and a change-over to drawing large-sized lumps of ore (up to 600-800 mm) secondary breaking is done in crushers installed in underground chambers near the shaft or the mouth of an adit. In this case use is made of high-capacity scrapers, large-sized chutes and rugged high-capacity cars. Drawing of large-sized ore requires replacing ordinary arc gate chutes (Fig. 161) with finger or chain gate chutes (Fig. 162). Jaw crushers

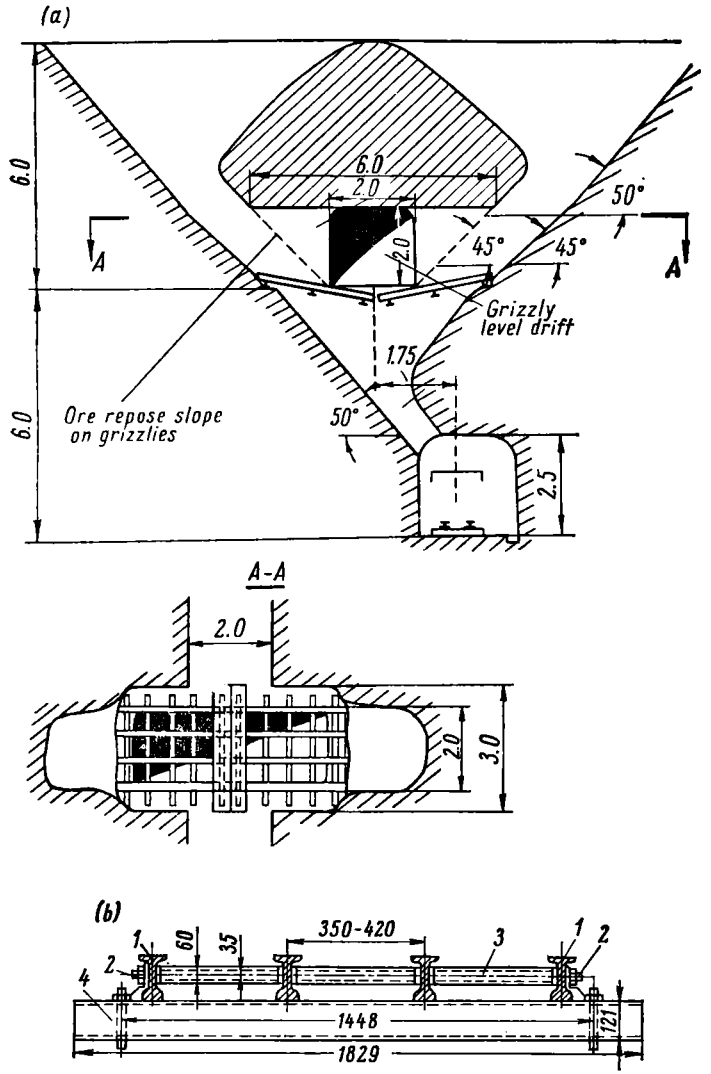


Fig. 160. Layout of a secondary-breaking-and-grizzly chamber

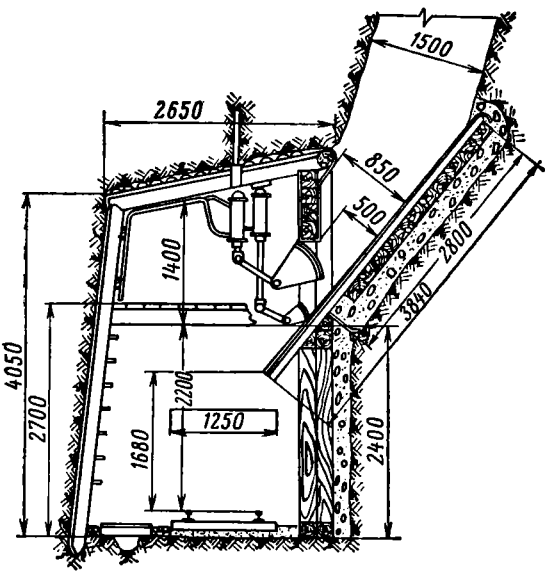


Fig. 161. Draw-chute with an arc gate

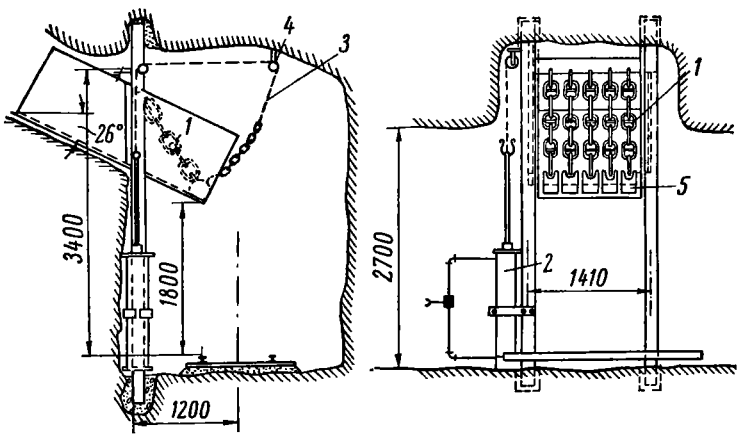


Fig. 162. Chain gate of a draw-chute:
1—anchor chain; *2*—air cylinder; *3*—rope; *4*—pulley; *5*—balancing weights

with charge openings measuring from 600 × 900 to 1,500 × 2,000 mm are usually employed for underground crushing. The capacity of heavy-duty crushers reaches 3,500 tons/hr. With drawing of hard ore in 600-800 mm lumps from several horizons onto a grizzly level the use of underground crushing plants is economically justified. At present underground crushers operate in a number of mines of the Krivoy Rog basin, in the Urals and in other regions. The use of larger lumps in the presence of a scraper level and the introduction of underground crushing plants for secondary breaking of hard ores (elimination of a special breaking level) represent one of the major trends in improving the technology of secondary breaking.

Among the measures aimed at improving secondary breaking is the use of thermit, mechanical devices (quarry-stone rams) and electrophysical procedures (high-frequency currents, etc.). Fragmentation of big lumps with thermit is due to the heat developing during its combustion and does not involve the scattering of ore fines or the formation of obnoxious gases and dust. Oversize lumps are fragmented into several pieces.

Fragmentation by means of pneumatic quarry-stone rams is practised at the Krivoy Rog mines. According to Eng. Delitsin they can be used successfully with ores of limited hardness containing a moderate number of oversize lumps (up to 22%). Among the promising methods of secondary breaking of ore are fragmentation of oversize with high-frequency currents (provided the ore possesses favourable thermophysical properties) and with the aid of hydraulic presses in the case of hard and very hard ores.

The application of novel crushing methods is of major practical importance and will appreciably improve the working conditions of miners and enhance the safety of operations (with manual and blasting methods of secondary breaking labour expenditure in secondary breaking and drawing of ore reaches 45-50% of the total labour consumed in stoping).

Major safety measures in sublevel stoping. These include the correct choice of the size of rooms, pillars and ceiling; proper illumination of rooms; workers engaged in open pockets should be tied to a safety rope. It is advisable to use closed pockets or fan-shaped rounds of blast-holes drilled from the sublevel.

The ventilation of a room causes no difficulties, since the air current passes from the haulage level workings via the room up to the top horizon and then towards the shaft. Blasting in sublevels should be done during the night shift, pillar caving when the mine is idle (days off, holidays).

Robbing of intervening and floor pillars in sublevel stoping. The methods used in recovering pillars may be divided into two groups: (1) robbing of pillars by top-slicing or sublevel caving with preli-

minary filling of the worked-out rooms with waste, and (2) pillar recovery with empty rooms. The first method is practised in mining very valuable ores (high losses and dilution of the ore are impermissible). The second method is employed in taking ores of low value.

With the first method the packing of the rooms is effected by gravity, through special workings excavated in the ceiling. The top-slicing and sublevel caving systems are discussed below (see classes of caving methods).

Pillar robbing with unfilled rooms may be done in the following order: (1) several cap and intervening pillars are blasted simultaneously, the caved ore is pulled out, the floor and sill pillars over the haulage roadway are recovered by sublevel caving underneath the caved waste overlying the discharged ore; (2) one or, more seldom, several cap pillars are blasted, the caved ore of these pillars is drawn out, the intervening pillars are robbed by top-slicing or, more often, by sublevel caving, the rooms being filled with the caved waste (after the caved cap pillar ore has been pulled out), following which the floor and the sill pillars over the haulage horizon are extracted by sublevel caving. The first order of extraction is employed rarely because of high losses and dilution of the ore.

The cap pillar can be blasted by using sectional-drill steel, longholing or concentrated chamber charges. Long-hole blasting is most widely used.

In drawing general conclusion as to room mining with sublevel stoping consideration should be given to the possibility of greatly improving it by increasing the level and individual sublevel intervals, the use of a flat or sloping floor, of fan-shaped rounds of long blast-holes, of a vibrating feeder for unloading the ore from the receiving funnels, trenches or the flat floor. The use of vibrating conveyers eliminates feed chutes on the haulage horizon, cuts down the amount of subsidiary development work, increases the size of the drawn lumps to 1,000 mm and raises the delivery efficiency to 300 tons/hr (3-4 times more than with scraper haulage).

A step that may greatly contribute to the efficiency of the system under discussion is the use of longholing, sloping floors and self-propelled equipment. The floor of the room is then specially prepared for the operation of mobile loading machines (Fig. 163). The output of one machine reaches 150 tons/hr and more.

According to estimates made by the State Institute for Projecting Enterprises of the Non-Ferrous Metal Industry the average output per shift per block provided with self-propelled equipment rises fourfold, the aggregate length of major and subsidiary development workings is reduced approximately in half and the time needed to prepare a block for stoping is reduced by 65 per cent. The number

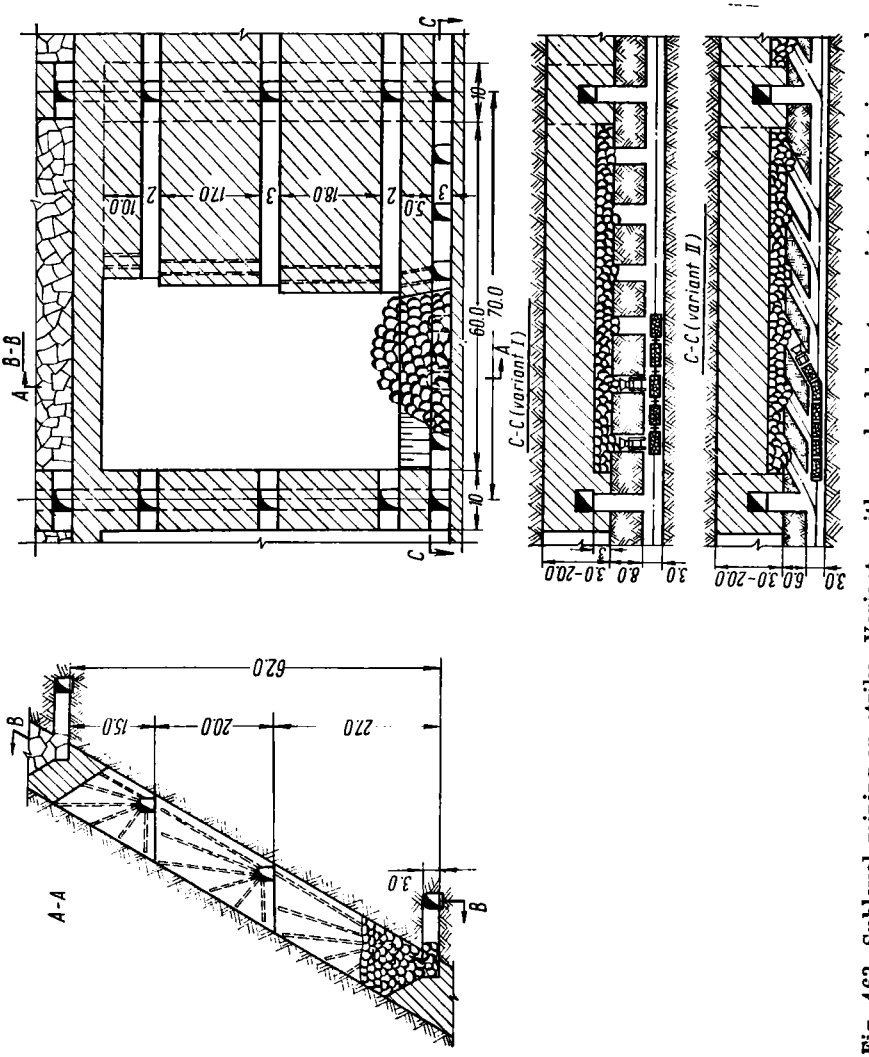


Fig. 163. Sublevel mining on strike. Variants with ore loaded onto an integrated train, or by shovelling machine into mine-cars

of blocks in the stage of development and stoping is cut down by 60 to 75 per cent.

The faceworkers' labour requirements per 1,000 tons of mined ore are decreased to 30 man-shifts instead of 180 with ordinary equipment.

The above information on the preparation of the room floor, the layout of drawing-off and undercutting levels, the drilling, charging

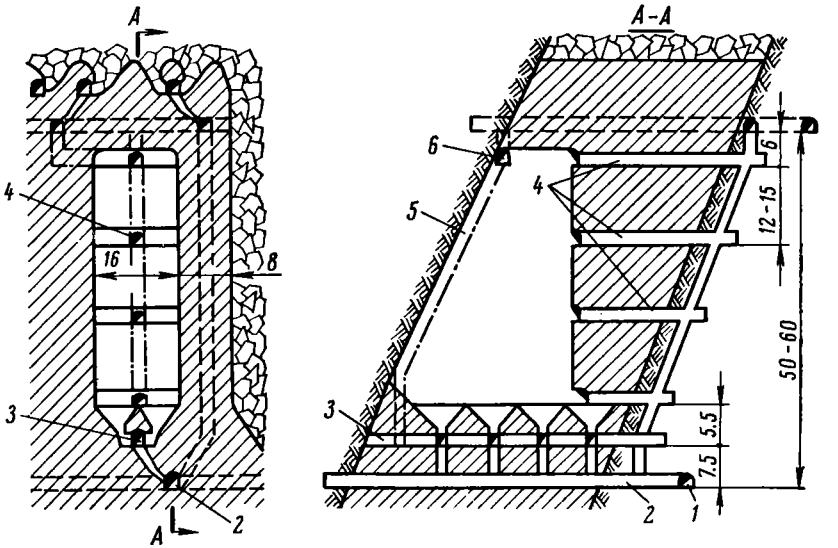


Fig. 164. Sublevel open-stope method of mining (by crossdrifts):
1—haulage drift; *2*—haulage crosscut; *3*—grizzly crosscut; *4*—sublevel crossdrifts; *5*—cut-out raise; *6*—ventilating breakthrough

and firing of blast-holes, the pulling of ore and secondary breaking refers to many mining methods involving long-hole breaking of ore, i.e. to most of the systems in classes I, II, and VII.

Sublevel breaking of ore from crossdrifts (sublevel crossdrift stoping). The system under consideration (Fig. 164) is used with ores and host rocks of considerable stability in working ore bodies more than 20-25 m thick dipping at an angle exceeding 45-50°, except for extremely thick ore bodies, which can be mined at any pitch angle.

The design characteristics of sublevel mining through crossdrifts are as follows: room width 12-18 m, thickness of intervening pillars not less than 6-8 m, room length up to 50 m, thickness of cap pillar not less than 6 m (0.4-0.6 times the room width).

With this system the total amount of the ore left in pillars is rather large.

With rooms located on strike the ratio of ore reserves in the room and intervening pillars is 6 : 1, with rooms running across the strike it declines to 2 : 1, 3 : 1 at best. In sublevel crossdrift mining, all difficulties associated with pillar recovery are much greater and so are the attending losses. The general order of mining in this variant and the methods for raising its efficiency are essentially the same as with ordinary sublevel stoping.

Modifications of sublevel crossdrift stoping (Fig. 165). These are used in ore bodies dipping at less than 45°, either with slushing the

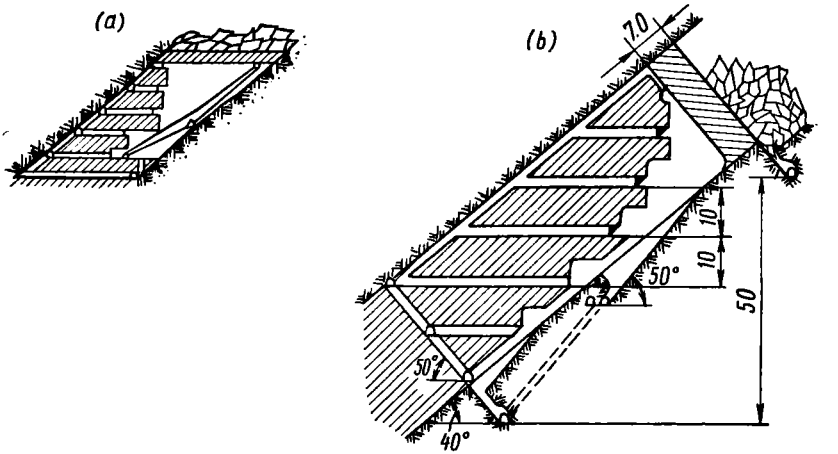


Fig. 165. Variants of sublevel mining of gently dipping deposits:
a—variant with scraper haulage; *b*—variant for thick deposits with broken ore drawn through stone pull-holes in the joot wall

ore along its footwall (Fig. 165*a*), or delivering it by gravity along draw holes from funnels cut in the footwall (Fig. 165*b*).

Advantages and shortcomings of ordinary or crossdrift sublevel stoping. The basic merits of this system include: safety of operations, since the workers always stay in mine openings (not in the stoped rooms); high labour productivity of facemen thanks to the possibility of using effective methods of breaking and drawing the ore; high intensity of stoping operations; the possibility of uniform delivery of the ore to the haulage level.

The chief shortcomings of the method are: difficult and slow recovery of pillars; a large volume of solid work; high losses and dilution of ore in robbing the pillars; impossibility of sorting the ore.

The technical and economic characteristics of the systems of ordinary and crossdrift sublevel stoping are listed in Table 20 (see p. 320).

Room Mining with Level-Wise Breaking of Ore through Long Holes (Horizon-and-Room Method of Mining)

Two basic variants of this method are in use with the ore broken in vertical and in horizontal slices.

Vertical slicing. The horizon is divided into blocks which are then taken out in two stages. In the first place rooms are extracted (on strike or across the strike, depending upon the thickness of the ore body), and in the second intervening pillars. Extraction of a room

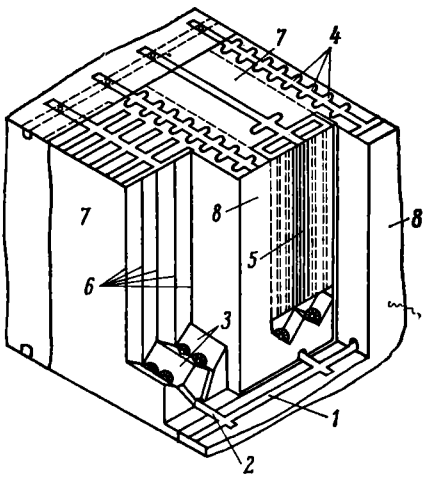


Fig. 166. Schematic representation of a variant with open-stope mining and breaking of ore throughout the whole level interval by drilling deep vertical holes:
1—drift; *2*—scraper crosscuts; *3*—trenches; *4*—drill drifts; *5*—holes made to form opening slot; *6*—holes made to break ore in room; *7*—solid ore in room; *8*—intervening pillar

is done by breaking the ore onto an opening slot by means of deep vertical blast-holes drilled with small drilling machines set up in workings of the top horizon.

A schematic representation of room mining across the strike is shown in Fig. 166. In the system with ore broken by vertical blast-holes the level interval is up to 70-80 m, the room width 15-25 m, and that of the intervening pillar 12-15 m.

The block is prepared for stoping by driving a set of workings excavated on haulage, scraper and undercutting floors, and through special drill workings excavated on the top horizon to accommodate small drilling units. Formerly, the cutting of the opening slot and the breaking of ore in the room were done by rotary drilling of long holes 115 mm in diameter.

Today this is done by using highly efficient rolling-cutter bits and rotary-percussion drilling machines. With such units blast-holes can be drilled upwards from the undercutting level, which eliminates the need for an upper drilling floor.

With this system intervening and floor pillars are robbed similarly to ordinary or crossdrift sublevel stoping.

Variant of breaking ore by horizontal slices. By way of example we shall consider a variant of the method with room mining on strike (Fig. 167). Preparation of the block together with the scraper level is clearly illustrated in the figure. The initial stage of stoping consists

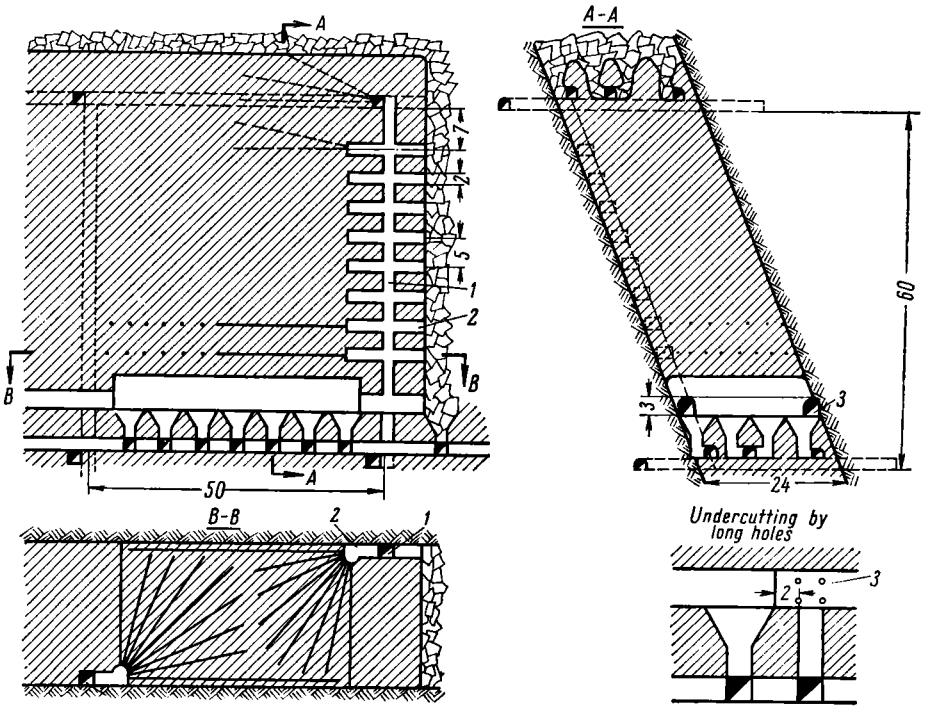


Fig. 167. Method of breaking ore by horizontal slices with fan-shaped rounds of deep holes

in undercutting of the room over its entire area. Undercutting by longholing from a stope drift (*3*) is highly efficient. Drill chambers (*2*) are excavated from raises (*1*) to facilitate longholing. The ore is broken in slices with the aid of fan-shaped rounds of blast-holes and is drawn via trenches (or funnels) cut in the floor of a room. The thickness of the broken ore slice is up to 5 m, depending upon the hardness of the ore and the parameters of drilling and blasting operations. The blast-holes are 25-30 m deep and run fan-wise. All the blast-holes in a slice are fired simultaneously, the broken ore being drawn shortly afterwards without prolonged storage in the room.

Cap and intervening pillars are taken out in much the same manner as in sublevel stoping.

The advantages of this variant are: exposure of the ceiling in the final period of room mining; better conditions for drilling horizontal blast-holes (as compared to upright ones); easy switching over to the ore shrinkage method. The drawbacks of the variant are: a large amount of solid work in connection with the drilling of horizontal blast-holes and difficulties in charging blast-holes.

When choosing between breaking ore in the rooms by vertical and horizontal blast-holes one should take into consideration the structure (jointing) of the solid ore in place and possible exposure areas in horizontal or vertical directions. At present preference is given to vertical blast-holes.

We do not consider here the rarely employed variant of the horizon-and-room method where horizontal slices up to 8 m thick are taken by firing chamber charges. Here, explosive charges of up to 400 kg are placed directly in sublevel workings driven from raises. Sublevel crosscuts are usually driven every 10 m and the charges spaced at 8-10 m. Owing to the difficulty of driving large sublevel openings and the complexity of charging, this variant is only applied in mining particularly hard ores. Due to the great achievements in drilling hard rocks by longholing, it is now replacing tunnel blasting with the use of chamber charges.

Breast-Stoping with Abandonment of Permanent Pillars

General. This method is employed in mining gently sloping ore bodies 1-2 to 4-6 m thick with stable host rocks. In an ore body of limited thickness the ore may have low stability.

With the system under review breast-stoping in an ore body is done either along the strike or up the raise, permanent pillars of lean ore or barren rock inclusions being abandoned in the course of stoping. The pillars are sited at random and are of different size. In the absence of poor ore or gangue inclusions in the solid ore pillars are sited in a regular fashion. In this case the room-and-pillar method of mining is used (see below).

In mining very valuable ores and when the deposit is less than 2 m thick artificial pillars made of concrete and supplementary support of the roof (cribs or packing) may be set up in place of natural ore pillars. Stowing is ordinarily employed in taking ore from thin producing areas with ripping of the back and driving of advance headings. The variants of breast-stoping with abandonment of permanent support pillars are distinguished by the mode of ore delivery or the shape of the stope face, i.e. scraper or conveyer haulage, delivery on rails to a gravity incline, or delivery on rails

to an ore-drawing chute made in country rocks, with rubber-tired self-propelled mine-cars to a drift, with an unstepped or underhand-shaped face.

Depending upon the angle of dip different modes of ore haulage are practised; thus rail transport and self-propelled mine-cars are used with a dip angle of up to 3°, scraper haulage for 0-30° and conveyer delivery for 0-15°.

Stope faces are shaped to conform with the thickness of the ore body, the technology and organization of drilling and blasting

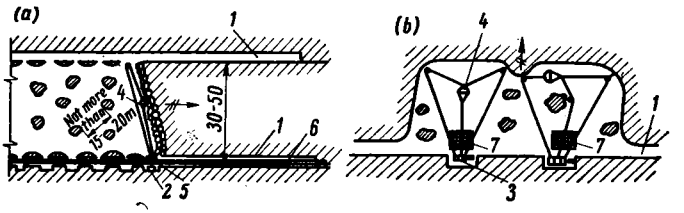


Fig. 168. Variants of breast-stoping with abandonment of casual permanent pillars (plan view):

a—stoping on strike with the use of double-drum scraper hoist; *b*—stoping to the raise with the use of triple-drum scraper hoists; *1*—productive drift; *2*—double-drum scraper hoist; *3*—triple-drum scraper hoist; *4*—scraper; *5*—mine-car loading scraper ramp; *6*—rail track; *7*—grizzly over chute

operations. Benchless stoping is commonly employed with ore bodies up to 3-4 m thick, underhand stoping with thicker ore bodies.

Breast-stoping with abandonment of natural pillars is mainly used at copper mines of Kazakhstan.

Variant of Benchless Stoping with Scraper Haulage of Ore. The variant of breast-stoping with abandonment of permanent natural pillars, the use of unstepped face and scrapper haulage of ore is shown in Fig. 168. It is practised in ore bodies not exceeding 3-4 m in thickness. Large-scale application of scraper haulage with this variant is due to the simplicity of work organization and the possibility of working at different dip angles. The effectiveness of scraper haulage can be increased appreciably by using the latest technical achievements (powerful triple-drum scraper hoists providing for a broad loading area without frequent shifting of rollers; remote control of the scraper hoist from the working stope).

The basic shortcoming of scraper haulage is the limited distance, which usually does not exceed 50 m. Therefore with great haulage distances conveyer transport or self-propelled mine-cars should be used; this, however, is often difficult to realize in the field because of the unsuitable dip angle and the need to employ special loading machines. The choice of a particular type of transport is made only after pertinent technical and economic calculations have been performed.

In this variant development work includes the drivage of productive drifts in ore with a lead of 30-50 m in respect to the production face line. Small chambers are cut from the drift to accommodate scraper hoists. Stoping is done with a continuous face on strike or up the raise (Fig. 168), abandoning pillars, usually randomly arranged. The area of these pillars comprises from 10 to 20% of the entire ore-bearing area. After the additional robbing of pillars (prior to the final reclamation of productive units) the aggregate pillar area may be cut down to 10%. Additional pillaring (second recovery)

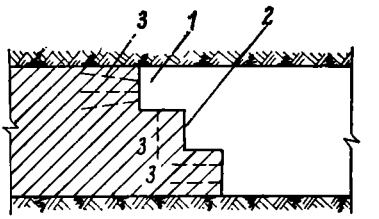


Fig. 169. Arrangement of blast-holes in underhand stoping (side elevation):
1—advance heading; *2*—bench; *3*—blast-holes

should be done from the boundaries of extraction areas, strictly observing the safety rules; for instance, in pillar recovery the piezoelectric devices described above may be used.

The ore is broken down concurrently along the whole face line or in its individual sections.

Variant of Breast-Stoping with Underhand Extraction and Scraper Haulage of Ore. With ore bodies exceeding 3-4 m in thickness breast stoping with underhand extraction is employed. In this variant an advance heading 2-2.5 m high is pushed forward directly under the roof, being followed by extraction of benches measuring 2-3 m in height in descending order (Fig. 169). In the advance heading horizontal or gently sloping holes are drilled, and in the benches lying below, vertical down-holes. To separate the ore from the foot wall rock, horizontal or gently sloping holes are drilled in the bottom bench.

The advantages of breast-stoping are: a small amount of development work; simple design features; a considerable stoping area; abandonment of gangue or lean unsalable ore inclusions; the possibility of sorting the ore directly in the stope area and retaining uninventarized ore reserves for future extraction.

The major shortcomings of the method are: high ore losses in permanent support pillars and difficult haulage conditions because of the random arrangement of the pillars.

In mining ore with abandonment of randomly located pillars subsurface field exploration has to meet severe requirements. It must ensure early and detailed assaying for proper outlining of intri-

cately shaped pillars. An additional demand on the exploration of ore bodies worked by the method under discussion consists in the correct setting up of the bottom elevation mark of the ore body. With an irregular profile of the floor and difficulties in laying down r.r. tracks one should resort to stone development, driving lateral drifts at a certain elevation below the bottom of the ore body. The use of rubber-tired self-propelled mine-cars allows of productive development with heavy gradients of haulage tracks.

Room-and-Pillar Mining of Ore

General. The room-and-pillar method of mining differs from breast-stoping in that production rooms alternate with intervening pillars and there are panel drifts run in advance (Fig. 170).

The room-and-pillar system is employed with regular mineralization and extraction of horizontal, gently sloping and inclined ore bodies of different thickness with stable enclosing rocks. It is preferred, however, in working ore deposits from 3-5 to 10 m thick with a horizontal or slightly sloping mode of occurrence. In sloping ore bodies rooms are extracted with the ore entrained and thrown away from the breast of the stope by a shock wave, or else they are worked along the strike (practice adopted in mining at the Mirgalimsay lead-and-zinc mine) (Fig. 171).

In a very thick ore body intervening pillars are not recovered.

In deposits of limited thickness the abandoned pillars are robbed partly (every other pillar), with strict observance of the safety regulations. Owing to considerable losses of the mineral in permanent support pillars (from 10 to 15%) room-and-pillar systems are used in working deposits containing poor ores only. When taking valuable ores the abandoned pillars are robbed after the rooms have been packed with self-cementing fill. In this case the recovery of pillars actually represents second mining. The use of efficient self-propelled equipment is one of the most important measures in raising the efficiency of the room-and-pillar method. At present, this equipment is widely utilized in copper and polymetal mines of the Kazakh SSR (Dzhezkazgan and Mirgalimsay). The proportional share of the total ore production by the room-and-pillar and panel-and-pillar methods with the use of self-propelled equipment should be greatly increased. When operating diesel-driven mobile machines it is necessary to take measures for entrapping noxious gases and enhancing the supply of fresh air (in the mining practice of the USA and Canada 2-2.1 m^3/min of additional air is delivered per each h. p. of diesel engines operating in underground workings).

Variants of the room-and-pillar method are described below.

Variant of the method with transverse underhand stoping of ore. This variant with a flat ore body (lense) is illustrated in Fig. 170. It is partly used in working the Dzhezkazgan deposit and some other occurrences.

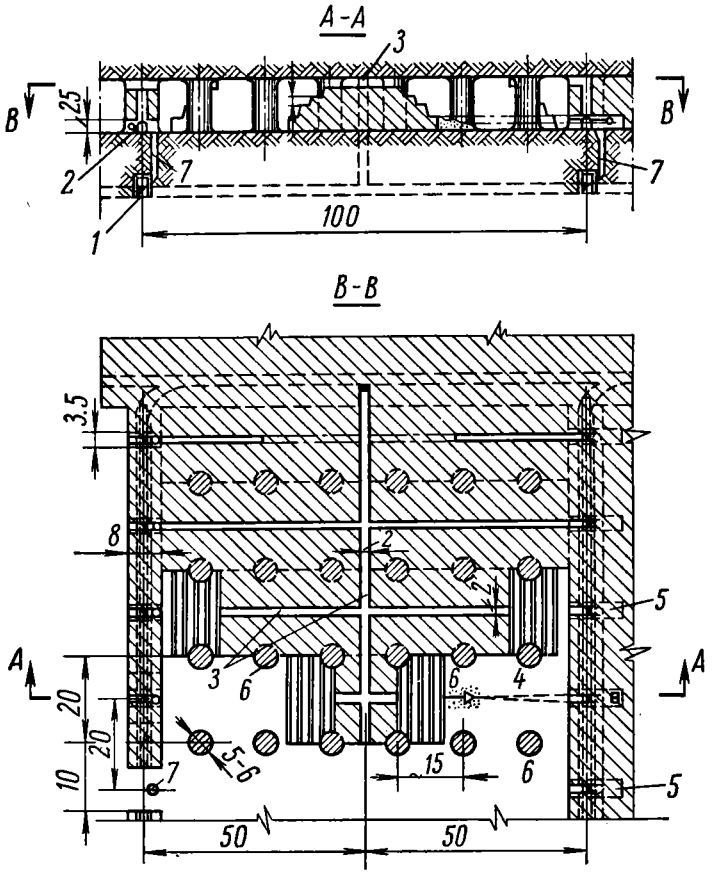


Fig. 170. Room-and-pillar method of mining with transverse underhand stoping of ore:
1—lateral panel drift; *2*—productive drift; *3*—airway; *4*—room; *5*—scraper hoist chamber; *6*—regularly arranged pillars; *7*—ore chute

The copper ore of the Dzhezkazgan deposit is represented by stable mineralized sandstones, the host rocks by stable sandstones. The coefficient of hardness of the ore and enclosing rocks is $f = 8$-12 according to Prof. M. Protodyakonov's scale. The ore in rooms is taken out by underhand stoping with abandonment of permanent

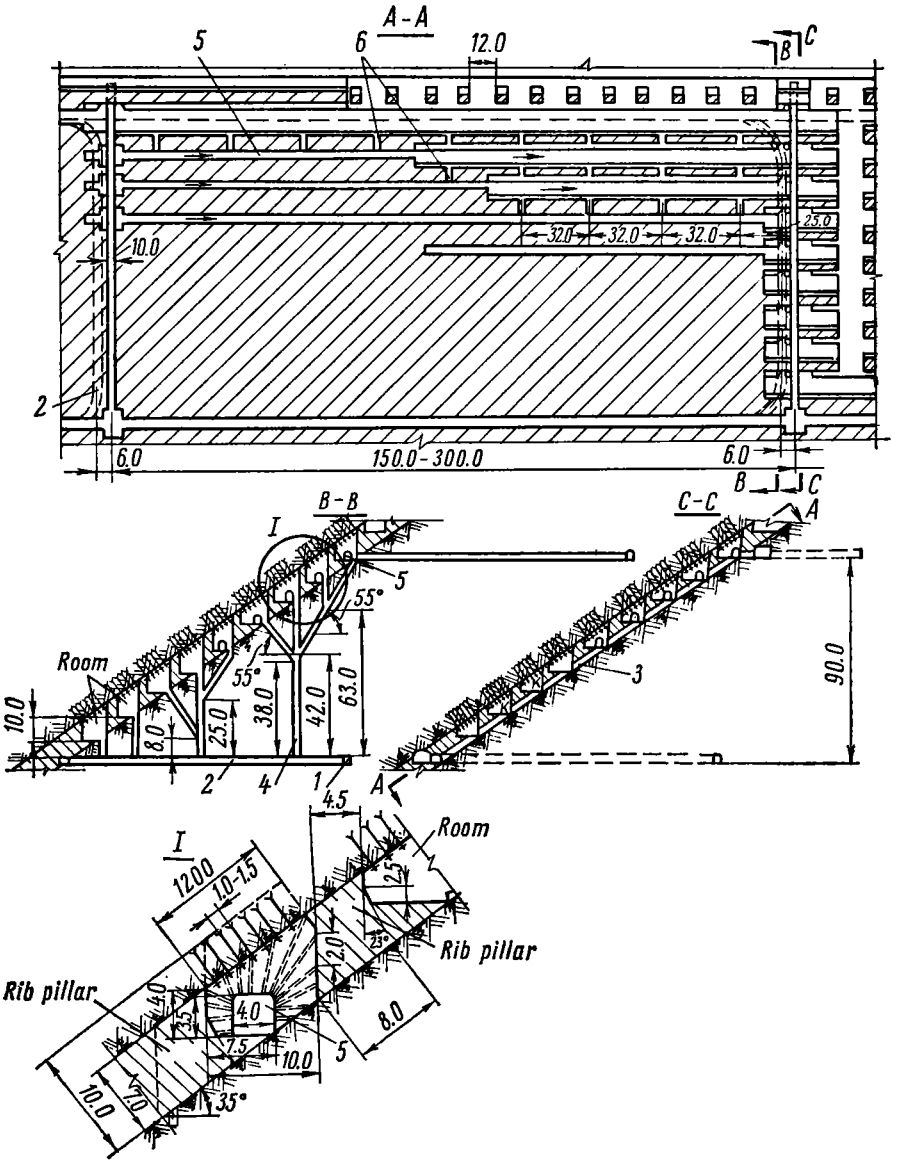


Fig. 171. Room-and-pillar system with stoping in rooms on strike and the use of self-propelled equipment:

1—lateral drift; *2*—crosscut; *3*—incline; *4*—ore-chute; *5*—stop drift running along the length of the room; *6*—ventilating breakthrough

natural support pillars arranged in a rectangular network. A panel is prepared for extraction by driving panel workings and ore-passes in subjacent country rocks, with airways excavated in the roof of the ore body.

The frequently changing elevation marks of the lense bottom under the conditions attending the working of this deposit rule out productive development with drift tracks having permissible up- and down-grades (with a regularly occurring ore body bottom, productive development would have been more efficient).

Panel openings are spaced at a distance of 100 m (two-way slushing with a maximal haulage distance of up to 50 m). Short raises with ladderway and ore-pass compartments are cut every 20 m in front of individual rooms from panel openings down to the bottom of the ore body. Small chambers are excavated from these raises to accommodate scraper hoists. A drift is run in the ore to ensure communication between production rooms. In order to provide adequate ventilation airways connected with an upcast shaft are driven in the roof of the ore body.

Extraction of ore in the rooms is started by excavating an opening slot, which is worked out by the overhand method. Once the opening slot has been formed an advance bench is started near the roof of the room, this being followed by normal development of underhand stope benching in the room. The room is worked out in the direction from the reef drifts towards the centre. Pillars are lined up in regular rows at the room boundary, their diameter being 5-6 m. They are arranged at a distance up to 15 m from each other, parallel to the room axis. The rooms are 20 m wide, the benches being up to 2.4 m high. Broken ore remaining on the benches is shovelled down to the floor of the room and then slushed into ore chutes. The variant described was widely used in pre-war years. Later on, the variant of the room-and-pillar method with overhand stoping and breaking of ore by firing shallow holes was successfully used at copper mines of Kazakhstan. This variant yielded better results. Other methods involving breaking of ore by firing shallow holes have also been employed at those mines, but we shall not dwell on them since they are less promising as compared with the variants involving the breaking of ore by means of long holes and the use of self-propelled equipment (see below).

Highly effective variants of the room-and-pillar method of mining. These include modifications with the breaking of ore by means of long holes, the use of mobile mechanical equipment and the hauling of ore by explosion shock waves. Below we consider individual variants of the room-and-pillar method.

Variant of the room-and-pillar method with breaking of ore by firing vertical long holes. With this variant stoping is effected in two stages

(Fig. 172): the first involves undercutting the room under its roof (with a deposit exceeding 10 m in thickness the room is undercut from below as well), and the second the breaking of ore by means of long holes drilled by machines from the top undercut stopes. This variant is used today at the Dzhezkazgan copper mines (Kazakh SSR).

The technical and economic characteristics of this variant are higher than those of the ordinary variants of the room-and-pillar system. With scraper haulage the output per faceworker was raised to 20-25 tons/shift. A change-over to breaking of ore into 700-800 mm lumps with the use of underground crushers, self-propelled drilling

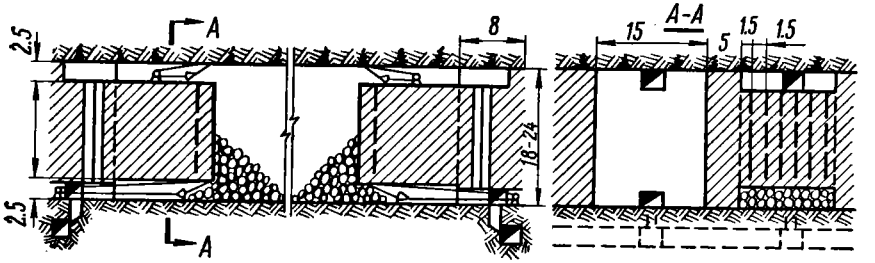


Fig. 172. Variant of room-and-pillar method with ore broken by firing vertical holes

and transport equipment may materially increase the effectiveness of this variant. Its substantial drawback is a strong seismic effect of exploding charges upon the abandoned intervening pillars, which results in their premature destruction.

Variant of the room-and-pillar method with self-propelled equipment and rooms on strike. This variant (Fig. 171) is used in the working of dipping deposits. In this method horizons up to 90-100 m high are extracted in blocks 200-300 m long. In each block inclines are arranged every 200-300 m in the bottom of the ore body which are provided with incline landings and hoists for lowering and hoisting self-propelled equipment into rooms arranged on strike. The mined ore is passed through a series of ore-chutes cut in the country rock.

A variant of the system with diagonal (sloping) ramps between rows of rooms is also employed. The rooms are made 10 m wide and the intervening pillars 4 m wide. The work in the rooms is done with a set of highly efficient Soviet equipment which includes self-propelled drilling units incorporating two percussive-rotary machines, mobile continuous power loaders, self-propelled mine-cars, self-propelled roof-inspection and bolting carriages (roof-pinning jumboes) (Figs. 173, 174, 175), mechanized pneumatic

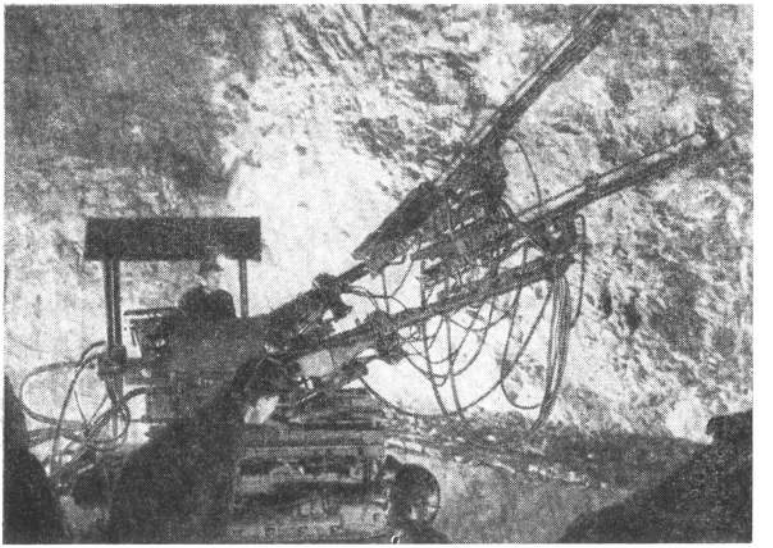


Fig. 173. Drilling of holes in a stope with a mobile СБУ-2 type long-feed drill jumbo at the Mirgalimsay mine (USSR)

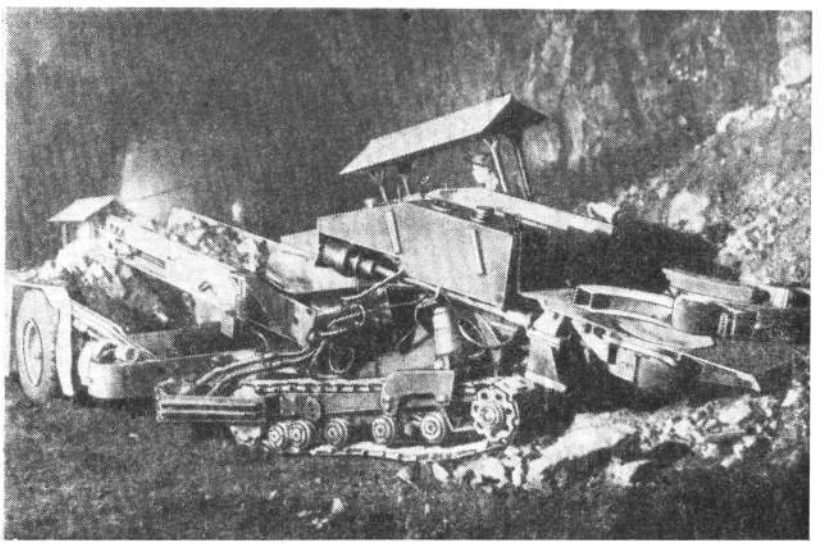


Fig. 174. Loading of ore at the face into a self-propelled shuttle car with a ПНБ-3к type mobile loader at the Mirgalimsay mine (USSR)

hole-charging devices and special sets of upraising machines. This machinery has appreciably increased the technical and economic efficiency and concentration of mining operations. The yield from

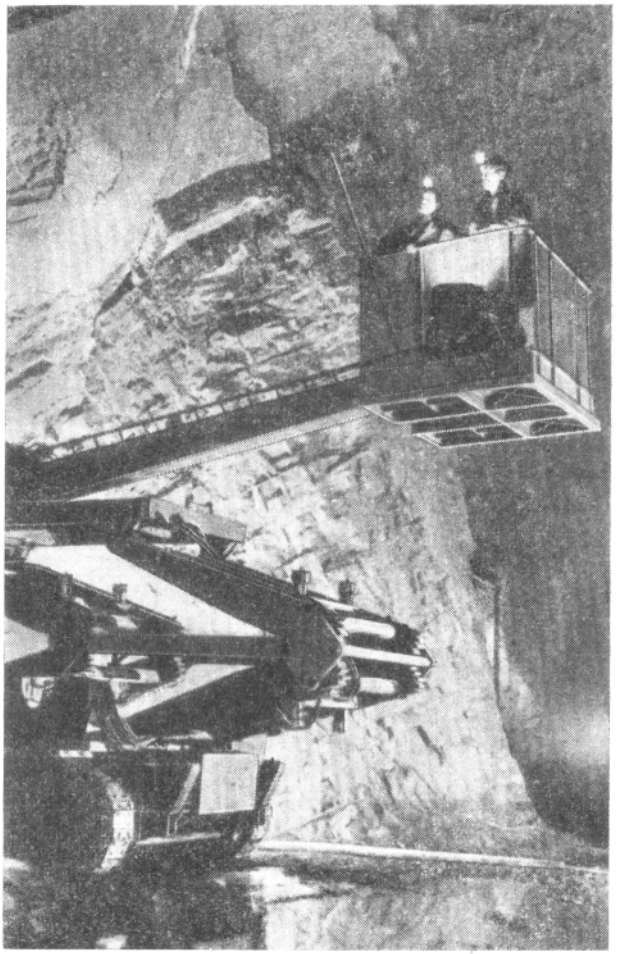


Fig. 175. Roof bolting in a room from a KCO-12 type mobile roof-pinning jumbo at the Mirgalimsay mine (USSR)

a block has risen from 4-5 to 30-60 thousand tons per month, the output per faceworker per shift has reached 50-60 tons (approximately a twofold increase). In individual instances these sets yielded over

60 thousand tons per month with an output per faceworker per shift as high as 100 tons.

Variant of room-and-pillar method with ore hauled by explosion shock waves. Owing to the difficulty of stoping in deposits dipping at 35-55° with the back composed of insufficiently stable rocks individual sections of the Mirgalimsay deposit are successfully worked by a system involving ore haulage by means of explosion shock waves (Fig. 176). This method has lately been introduced in a number of other mines of the USSR. A level with a vertical interval of 25-60 m is mined through open rooms 15-25 m wide with abandonment of pillars 8 m wide. Every other pillar is subsequently recovered by using the method employed for the extraction of rooms. In the lower portion of the level a floor is prepared and a cutoff stope is driven across the entire thickness of the deposit, normally to its dip. Stoping operations proceed up the raise. The ore is broken down by firing fan-shaped rounds of blast-holes drilled from the raises. The floor is shaped so that the ore can be passed onto the haulageway via a draw-hole or discharged with the aid of a vibro-conveyer. With highly efficient drilling and blasting and the possibility of servicing 2 adjacent rooms such a vibro-conveyer may be very efficient. Researchers at the Skochinsky Institute of Mining have established that the distance to which the ore is thrown by an explosion shock wave is contingent upon the dip angle of the deposit, the initial flying speed, the cross-section of ore lumps and the coefficient of friction between the ore and the floor. Pertinent calculations have proved, for instance, that at a dip angle of 15°, a coefficient of friction of 1.0, a lump size of 100 mm, and an initial flying speed of 5 m/sec, the shock-wave haul distance is 10.4 m, the respective figures for initial velocities of 10 m/sec and 15 m/sec being 23.1 and 37.9 m. At pitch angles less than 15° the flying distance diminishes, whereas with dip angles exceeding 15° it increases. The flying distance also varies with the coefficient of friction. The absence of workers in the stope area enables wider rooms to be excavated.

Panel-and-room method of mining. In the Dzhezkazgan copper ore deposit mining is done by the panel-and-room method with extensive use of self-propelled equipment. A panel is taken out in any one direction with abandonment of permanent natural support pillars. The type of self-propelled machines used depends upon the thickness of the ore body. In the practice of the Dzhezkazgan mine use is made of the following sets of self-propelled machinery:

(1) in ore bodies 6-12 m thick: СБУ-4 drilling unit, ЭП-1 underground power shovel, diesel-driven mine-car, diesel-driven crawler-mounted bulldozer, СП-12 mobile walling scaffold for assembly and bolting of roof suspension support;

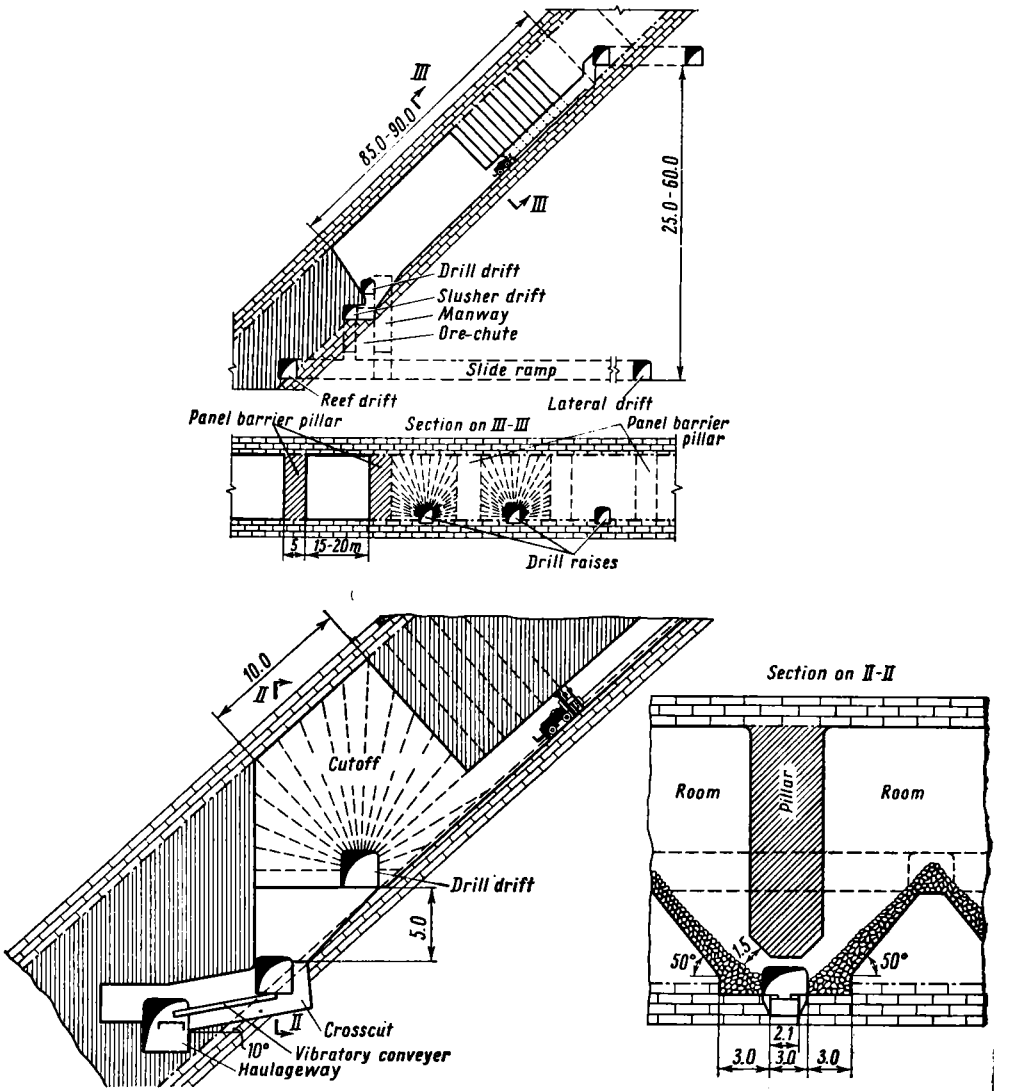


Fig. 176. Mining method with the ore hauled by explosion shock waves at the Mirgalimsay mine (USSR):

a—variant with ore passed down to haulage level through ore-chute; *b*—variant with ore discharged to haulage level with the aid of vibrating conveyer

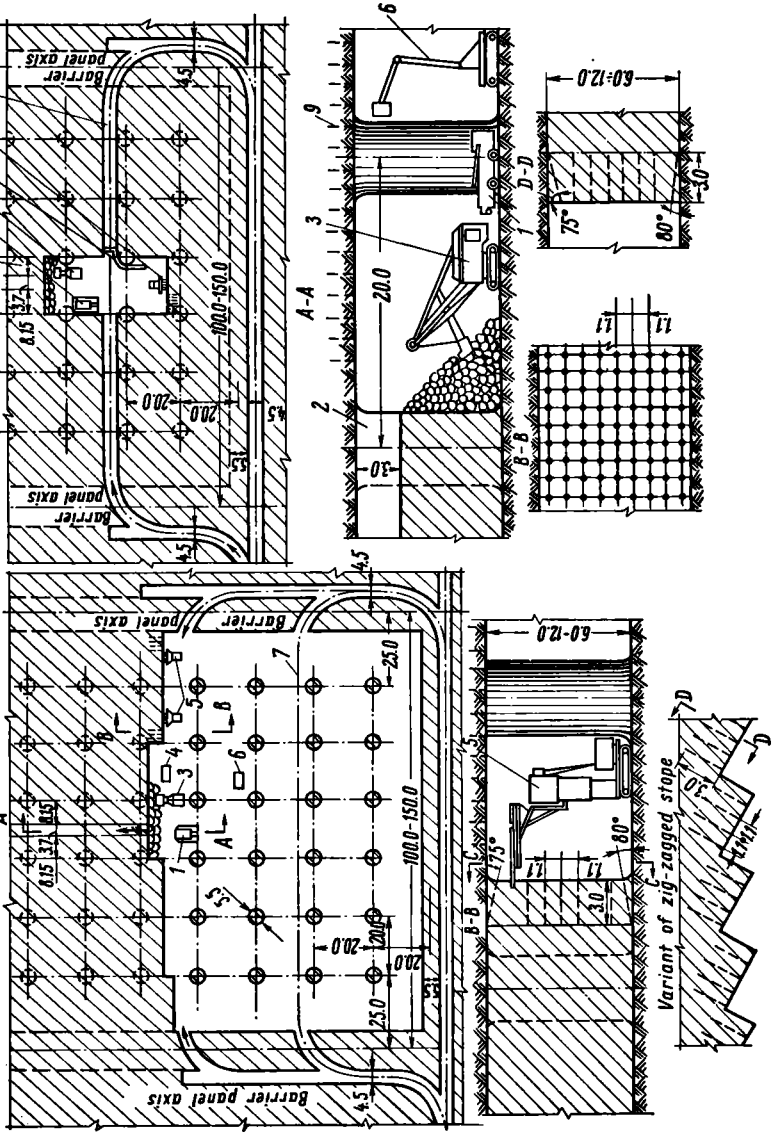


Fig. 177. Panel-and-pillar method of mining:
1—trolley-and-cable car; 2—vent drift; 3—power shovel; 4—bulldozer; 5—drill carriage; 6—mobile roof-inspection scaffold; 7—trolley wire; 8—fan; 9—suspension roof support

Table 20

Mining methods	Coef. of recovery	Dilution factor	Timber consumption, m^3/ton	Explosives consumption, kg/t	Output per face worker/shift, tons
Overhand stoping with stull timbering	0.94-0.95	0.10-0.15, depending upon ore body thickness and gangue inclusions (with ore body thickness of at least 0.8 m)	0.03-0.04	0.5-0.6	4-6
With stull timbering and skeleton shrinkage of ore	0.94-0.95	0.10-0.15, depending upon ore body thickness and gangue inclusions (with ore body thickness of at least 0.8)	0.01-0.015	0.35-0.50	8-12
Sublevel stoping with subdrifts or crosscuts (extraction of rooms)	0.95-0.97	Depends upon gangue inclusions (in practice often from 0.05 to 0.10)	—	0.3-0.4	30-40
Breast-stoping with abandonment of pillars	0.88-0.90	Depends upon gangue inclusions (in practice often from 0.05 to 0.15)	—	0.4-0.5	10-15
Room-and-pillar mining: underhand stoping	0.88-0.90	Depends upon gangue inclusions (in practice often from 0.05 to 0.15)	—	0.4-0.5	10-15
longholing	0.85-0.88	Depends upon gangue inclusions (in practice often from 0.1 to 0.15)	—	0.30-0.40	20-25 (up to 50-70 with self-propelled equipment)
Horizon-room method	0.92-0.95	0.10-0.15	—	0.3-0.5	40-50 (up to 70-80 with self-propelled equipment)

Note:
1. In this and subsequent tables the figures imply a unit weight of ore averaging 3 tons/m^3.
2. Figures for sublevel stoping with subdrifts or crosscuts and for the horizon-room method are given with special reference to room stoping. To arrive at average figures for extraction of a block (the room and all pillars) it is necessary to take into account the proportional share of the systems used in taking out ore from individual portions of the block and the characteristics of these methods. Figures for pillaring should be somewhat lower in view of the difficulties encountered during their recovery.
3. The indicated coefficient of recovery in breast-stoping and room-and-pillar mining allows for partial robbing of pillars in secondary mining of panels.
4. The dilution factor should be determined in each individual case with special reference to the existing mining conditions.
5. Table 20 does not take into account the working of occurrences containing low-value minerals (building materials, rock salt, etc.). Such deposits are mined by room methods with considerable reserves of the mineral left in permanent support pillars (up to 40-50%).

(2) for thicknesses of 3.5-6 m: СБУ-2 drilling unit, ПНБ-3k gathering arm loader, ВСД-10 self-propelled mine-car, БПД underground electrically driven bulldozer, САКК-2-2,000 self-propelled roof-pinning jumbo;

(3) for thicknesses of 1.5-3.5 m: СБУ-2 drilling unit, ГСС-1 mobile scraper loader, ВСД self-propelled mine-car, БПД underground electrically driven bulldozer. Figure 177 exemplifies panel-and-room mining in a deposit 6-12 m thick with the use of power shovels, self-propelled mine-cars and drill carriages. The width of an extractable panel is between 100 and 150 m; the barrier pillars are 13-27 m wide, depending upon the mining depth (100-275 m). Rock bolting is arranged on the following network pattern: from 1.0 × 1.0 m in red and grey shales to 2.0 × 2.0 in red dense sandstones. Drilling of holes up to 3 m deep is done with drill carriages.

The basic technical and economic characteristics of the open-stope mining methods are listed in Table 20. These figures, as well as those given in other tables covering other classes of mining methods, are tentative and can only be used in approximate calculations.

The technical and economic characteristic of the mining methods listed in Table 20 and other tables should not be regarded as permanent. In view of the vigorous development of technology and continuous improvement in the organization of mining the corresponding corrections should always be introduced.

5-3. SHRINKAGE MINING WITH ORE STORED IN THE STOPE AREA (CLASS II)

General

Methods of mining involving shrinkage of ore differ from other systems in that the major part of the broken ore is left in the stope area for a long period. After each caving of the ore, a certain amount of it is drawn free, so that sufficient space is left between the surface of the broken ore and the back of the stope to enable the subsequent operations to be carried out.

Shrinkage methods of mining are highly efficient and the possibility of their application should be given considerable attention. The main condition for their use is the stability of the ore and enclosing country rocks sufficient to make possible the breaking of ore without timbering and the pulling of ore without the caving and spalling of the host rocks. The other prerequisites are the absence of barren rock inclusions in the ore body, the uniformity of the ore (since its sorting in the stope area is impossible), the presence of regular contacts between the ore body and the country host rocks,

and the absence of compacting and self-ignition of the ore during long-term storage in the shrinkage area. The dip angle of the ore body should not be less than 50-55°, except in very thick deposits, which can be mined by shrinkage methods at smaller angles of pitch. Shrinkage mining is applicable in ore bodies of different thickness—from very thin to extremely thick ones.

Shrinkage stoping of ore is employed in the working of gold and non-ferrous metal occurrences. It is rarely used in mining iron ores.

The proportional share of shrinkage methods in the mining of lode deposits of non-ferrous metals and gold is now about 60%. There are reasons to expect a further increase in this figure.

In shrinkage mining the ore can be broken in the usual way—by blasting shallow holes (without a grizzly level) or in bulk with long holes and a grizzly or scraper level. The former method is chiefly used in the working of deposits of limited thickness, the latter in the mining of thick occurrences.

Class II of the mining methods includes the following methods applied in the working of Soviet ore deposits:

1. Mining in blocks on strike with the ore broken out by firing blast-holes.
2. Room stoping along or across the strike with the ore broken out either by firing shallow holes or by longholing.
3. Breast-stoping of the ore with the drilling of ordinary holes.

Mining in Blocks on Strike with the Ore Broken Out by Firing Blast-Holes

This method (Figure 178 represents one of its variants) is practised under conditions common to all systems of shrinkage mining. An additional point is that the thickness of the ore body should be from 0.3-0.5 to 2-3 m. In a lode 0.3-0.8 m thick extraction is accompanied by taking out enclosing country rocks (in a lode less than 0.5 m thick one should draw economic comparison with selective extraction and backfilling). In the system discussed a 40-50 m high level is divided into blocks 50-60 m long. No pillars are usually left above the drift and at raises. A sill pillar is abandoned in taking lean ore, then the drift is pushed forward without timbering and short ore-chutes are driven for the drawing of broken ore. Preparation of the block for stoping includes the driving of double-compartment raises and a haulageway. The latter has a single track and is supported with frame sets (Fig. 178) or stulls. In the former case purlins 20-25 cm in diameter are laid down on cross-beams of the frame sets along the hanging and foot walls, and braces are wedged between them every 1.5-2 m. The purlins are covered with short round lagging, holes being left in it for chutes.

When the back is supported with stull-pieces the latter are set up in a single or double row. Double-row timbering is put up in the case of partial picking of waste lumps from the ore pulled out, and in this case simple draw-holes are made in the roof of the drift, with the ore loaded into mine-cars and members of the transverse flooring pushed aside (Fig. 179).

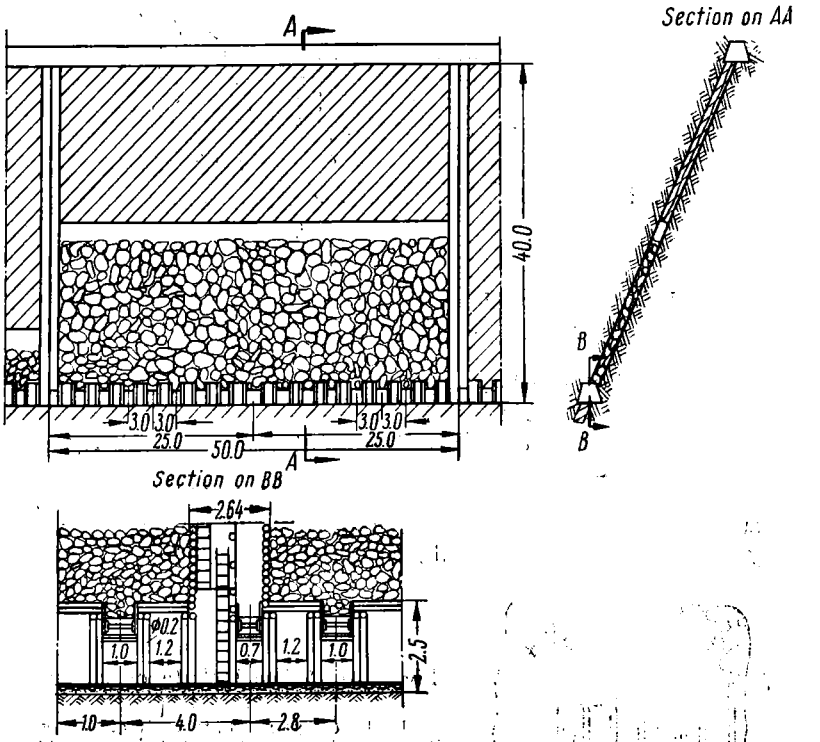


Fig. 178. Shrinkage stoping in blocks on strike with ore broken by firing shallow holes

On completion of development work the shrinkage stope is undercut; to do this a slice of ore 2 m high is taken out over the drift. To facilitate undercutting the level drift is ordinarily timbered, not during driftage, but right after undercutting of the shrinkage stope in such a way that the ore broken in the course of undercutting falls onto the floor of the drift. In undercutting, holes are drilled with stopers. Upon termination of undercutting and timbering of the haulageway winged chutes and draw-holes for pulling the ore (Fig. 180) are made. If the distance between the draw-holes is small

(less than 3 m) no winged chutes are made (Fig. 178). After the shrinkage stope has been undercut and draw-holes made backstoping or extraction with continuous faces is started (Fig. 178). Miners drill the solid ore in place with stopers, standing on the broken ore. The depth of the holes is limited and usually does not

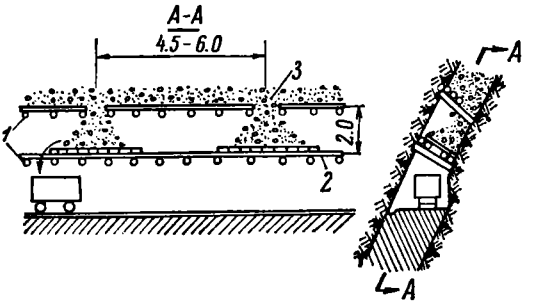


Fig. 179. Timbering with double-row stulls and chutes in the roof of the drift:
1—stulls; *2*—purlins; *3*—broken ore

exceed 2-2.5 m, i.e. the height of the slice to be broken. With closely spaced holes of such depth no secondary breaking of ore is needed. Oversize lumps are either fragmented or additionally blasted at the face.

In the course of extraction part of the ore (30-40%) is pulled out, the rest being left in the stope area (shrunk). The ore is drawn completely only after the entire block has been mined out.

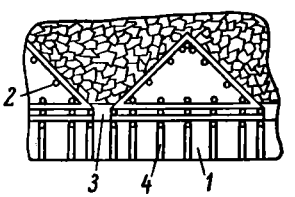


Fig. 180. Winged chute over haulageway:
1—haulageway; *2*—chute built of battered props and lagging; *3*—draw-hole; *4*—frame-set posts in drift

As indicated above, in shrinkage mining the grading of the ore is generally ruled out, but in working very thin lodes and with a distinct separation of the ore in two grades across the entire thickness of the lode such grading is feasible.

The ore in the block is then usually taken out in horizontal slices without dividing the block into benches, a flooring being laid down prior to extraction of one of the grades, with subsequent delivery of this grade to the ore-pass compartment of the raise. The other ore grade is stored in the shrink, and it is drawn off only after the entire block has been stoped out. The effectiveness of the system under

review may be appreciably increased by making an inclined floor and drawing the ore directly onto the haulageway (Fig. 181). In this variant the haulageway is run in the country rock, from which cross-drifts (stub entries) 7-10 m long are driven. The ore in the cross-drifts is loaded with a loading machine into a storage-train or separate mine-cars. With chuteless drawing of ore and an inclined floor the design characteristics of the method are greatly simplified. The output per faceman can be substantially increased as compared with the ordinary variant of the system.

Radical improvement of shrinkage mining in working thin deposits. The investigations carried on for many years by the Skochinsky Institute of Mining and the field experience gained by the "Khrustalny" and other mines have shown that the efficiency of shrinkage stoping of ore can be greatly increased through radical improvement of the system and changing the entire technology of stoping. The following measures have served as a basis for the novel technology and organization of stoping operations (Fig. 182).

1. Breaking of ore with holes of reduced diameter (34-36 mm in lieu of the ordinary 44-46 mm), this raising the drilling rate of the existing equipment by 70-80%.

2. The use of new high-percussion-rate stopers for holes of smaller diameters which have increased the drilling rate by 20%.

3. Substitution of cross bits with slotted cutter blades for ordinary supplementary tipped ones, which has raised the drilling rate in fissured grounds by 15-20% (Fig. 182*d*).

4. Disposition of holes on a network pattern with an allowance for the physical and mechanical properties of the ore. This has enabled the number of holes to be cut almost in half and the efficiency of breaking to be raised by 75-80%.

5. The use of a new explosive—detonite packed in cartridges of reduced diameter in conjunction with widely spaced small-diameter holes has ensured the required size-grading of ore, eliminating its overbreakage and reducing dilution with wall rocks.

6. Drawing of ore from shrinkage-stopes via continuous chute gates, thus securing uninterrupted and intensive ore pulling and reduction in labour expenditure almost by half (Fig. 182*c*).

7. Decreasing the scope and simplifying the excavation of temporary development workings in the block by dividing it into two independent semi-blocks (Fig. 182*a*).

8. Partial use of stull timbering inside shrinkage stopes which has broadened the application of the method in insufficiently stable host rocks.

9. Organizing an integrated team consisting of skilled faceworkers qualified in several different jobs, and operating under a cyclic schedule (Fig. 182*b*).

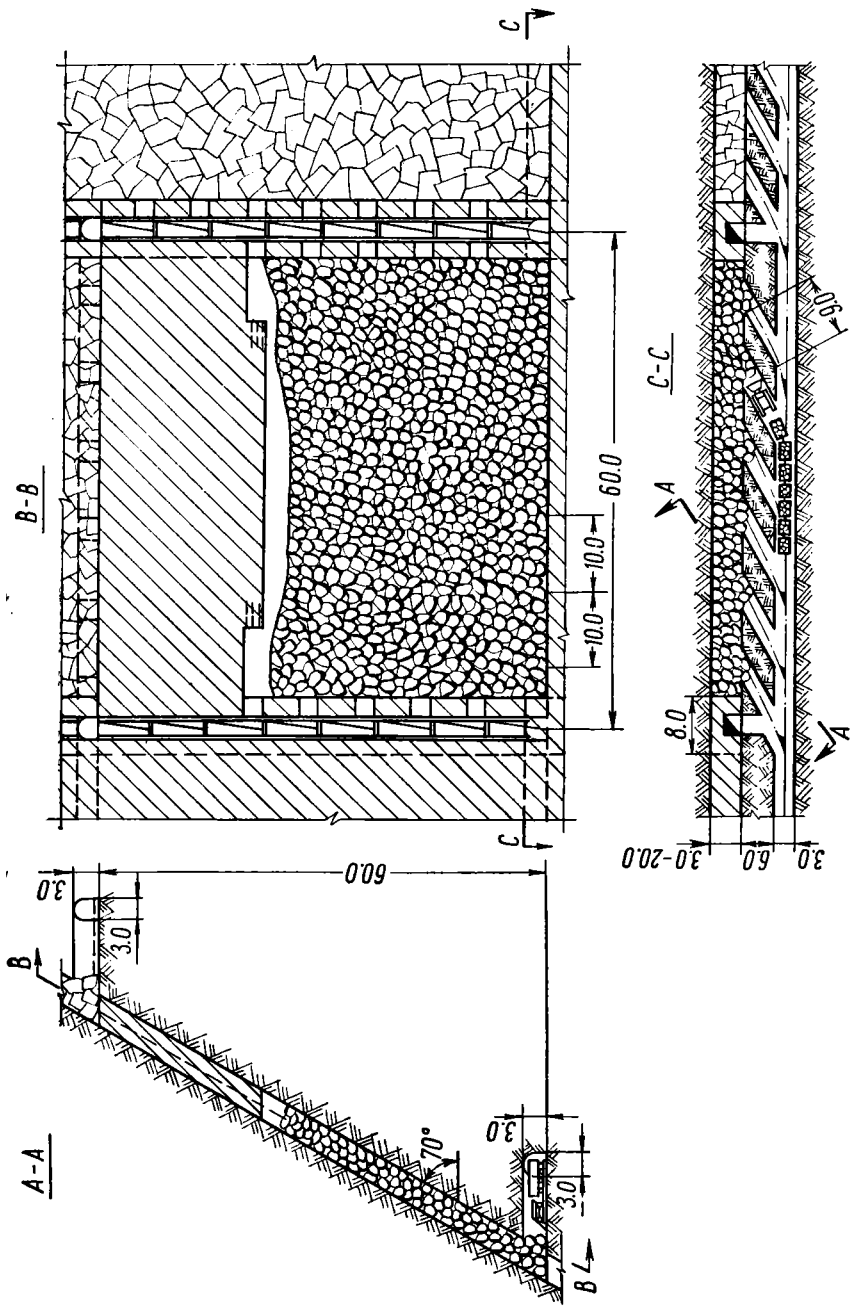


Fig. 181. Shrinkage stoping of ore along the strike. Variant involving loading of ore with a loading machine into a storage train or into an ordinary train pulled by an electric loco

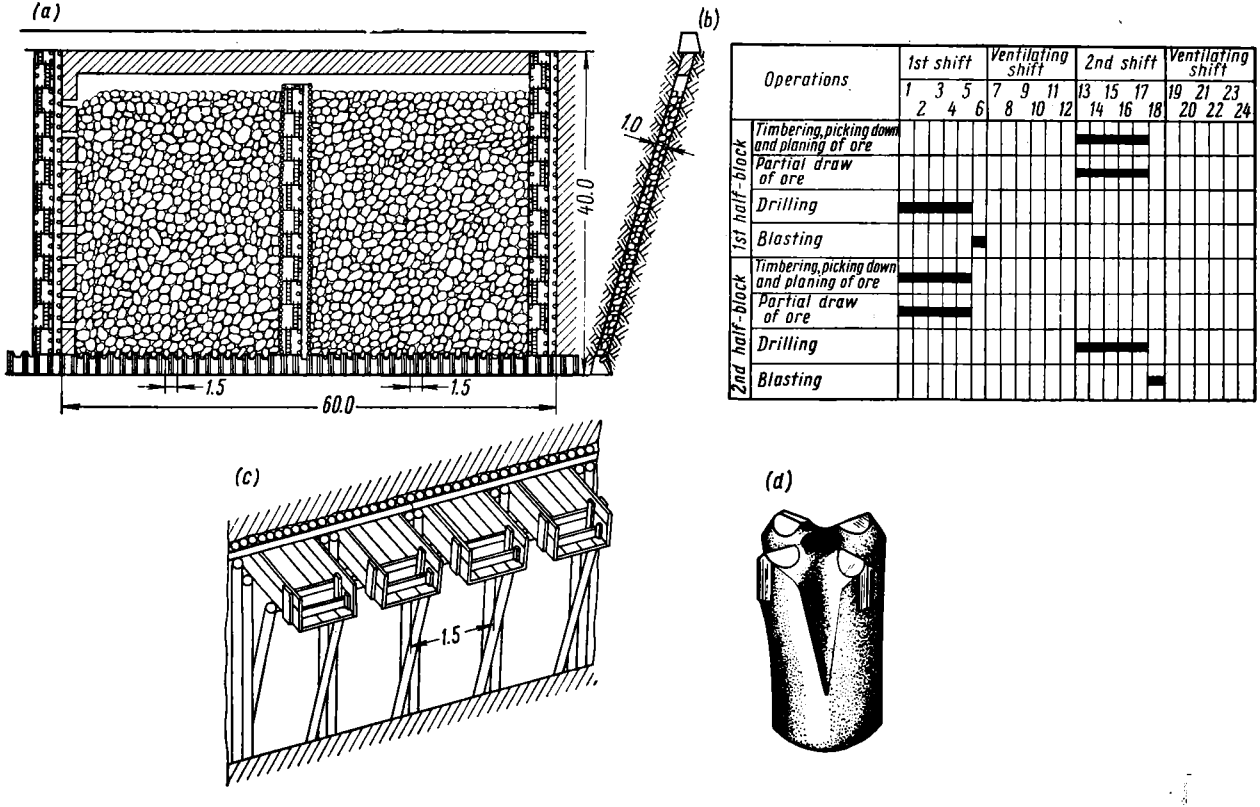


Fig. 182. Shrinkage stoping of ore at the "Khrustalny" mine

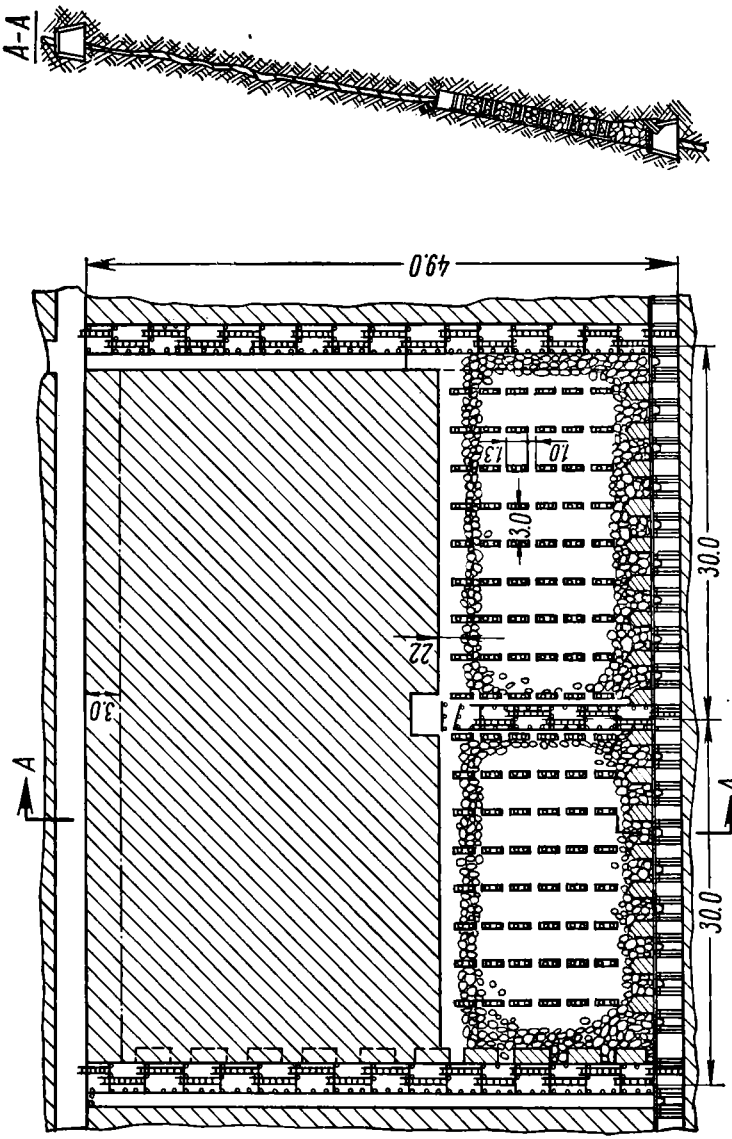


Fig. 183. Shrinkage and stull-set method at the "Davenda" mine

After the realization of the above-mentioned measures the "Khrustalny" mine had achieved remarkable results, such as an output of up to 10.5 cu m per faceman per shift, the rate of the face advance in a block up to 40 m per month (mining out of a block in one month). The experience gained by the "Khrustalny" mine is being passed on to other mining enterprises of the Soviet Union.

With a view to extending the use of shrinkage stoping of ore in insufficiently stable enclosing rocks the latter are held in place by means of side and roof bolting, the ore is taken out of a block through short shrinkage stopes (the block being divided into three shrinks, each 15-20 m long), or shrinkage stoping of ore is combined with the stull-set method, the stulls being arranged in regular rows (Fig. 183). The shrinkage and stull-floor method of mining has yielded good results at a number of mines in the USSR, the following rules being observed: the distance between the timber set rows should be at least 3-4 m and the rows should run along the ore-chute axis. Rock bolting is more effective when exfoliation occurs along the planes parallel to the plane of contact between the host rocks and the ore body. Wedge-type metal bolts 25 mm in diameter are arranged on a network pattern (1.5×1.5 to 2×2 m). The plug holes to accommodate the bolts are drilled with small jackhammers, using sectional drill steel and couplings 33 mm in diameter. With 25 mm bolts 36 mm plug holes are drilled. The wedge lock can withstand a load ranging from 5-6 tons (in soft grounds) to 9-10 tons (in hard rocks).

Shrinkage Mining of Ore by Rooms on Strike with Blasting of Shallow Holes

This method (Fig. 184) is applied in working ore bodies from 3-4 to 10-15 m thick (the general conditions for its use have been discussed above). The ore body is extracted by levels 40-60 m high. The level is broken up into individual producing sections (blocks) 40-60 m long. The length of the block should be correlated with permissible roof and side exposure areas in the room. Each block consists of a room and an intervening pillar 4 to 6-8 mm wide, depending on the ore body thickness, stability of the ore and the host rocks, and also on the contemplated method of pillar recovery.

Pillars 4-6 m high (depending upon the thickness of the ore body and the hardness of the ore) are usually left above and below the haulageway. The height of the floor pillar should be 0.4-0.6 times the room width. The height of the sill pillar is so stipulated that it could accommodate receiving funnels and ore chutes.

The ore is extracted in a definite sequence with regard to the mining of rooms and pillars in individual levels. Like in other room-mining

methods rib and cap pillars in the horizon should not be robbed until the overlying level has been fully mined and abandoned. It is very important that the room and intervening pillars in different levels be equal in width, with their axes strictly coinciding.

The development workings in the block consist of air- and haulageways, and raises arranged in rib pillars.

The subsidiary development workings include: manways driven from the raises every 4-6 m to provide communication with rooms, an undercutting drift (4-6 m above the haulageway), from which

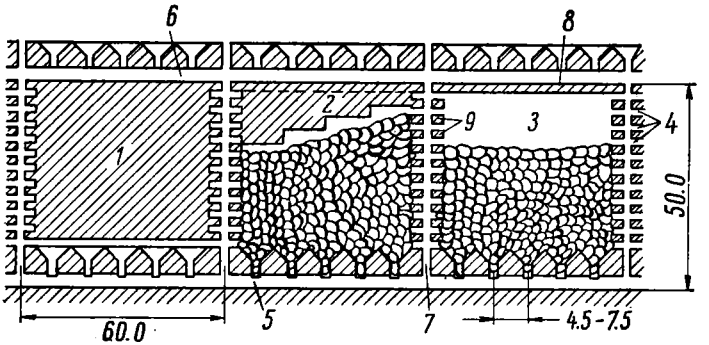


Fig. 184. Deployment of work in shrinkage stoping of ore by rooms on strike: *1*—room in the stage of completed development and solid work; *2*—room in the stage of shrinkage stoping; *3*—room in the stage of drawing the ore from the shrinkage stope; *4*—ladderways; *5*—haulage drift; *6*—vent drift; *7*—raise; *8*—ceiling; *9*—intervening pillar

undercutting is done throughout the entire room area, short orechutes spaced at 4-6 m, and receiving funnels for drawing the ore. The funnels are made by enlarging the upper portion of short orechutes.

Breaking of ore is started by drilling holes 2.5-4 m deep in the back of the room so as to form an overhand stope and ensure ample face room for drillers. Holes are made with drifter hammers or, more often, with stopers. In the case of partial drawing of broken ore from the shrinkage-stope all miners must leave it, since the stored ore may suddenly begin sinking in the course of drawing.

When performing stoping operations provision should be made for detailed inspection and picking down of the roof after blasting, the presence of at least two exits from the shrinkage stope via manways to the raises, free passage over the mass of the broken ore (the working space should have head room of not less than 1.5-2 m), and additional breaking of oversize lumps in the shrink.

The ore in a shrinkage-stope is taken out up to the ceiling. Where there are manways communicating with raises ventilation of the

stope areas does not present any difficulty. With long rooms a special ventilating raise is sometimes driven in the middle of the room. Intervening and level pillars are recovered in the same way as in the sublevel method of mining.

The technical and economic characteristics of the system are listed in Table 21, p. 335.

Shrinkage Mining of Ore in Rooms Across the Strike by the Short-Hole Method

This system finds application in conditions analogous to those typical of the just described method, in ore bodies exceeding 10-15 m in thickness.

The rooms are 8-12 m wide, their length corresponds to the thickness of the ore body, but usually does not exceed 50 m. With a very stable ore the width of the rooms may be increased to 15-20 m. The abandoned intervening pillars are 6-8 m thick, depending upon the contemplated mode of their robbing. The size of the pillars is increased when they are recovered by top-slicing and sublevel caving after the rooms have been filled.

The general sequence of development, subsidiary development and stoping is similar to that adopted in shrinkage mining of ore in rooms on strike. Development involves excavating a stone drift in the foot wall.

Raises are run from crossdrifts in intervening pillars and rooms. With rooms of considerable width raises in the pillar are driven every 20-25 m across the strike. To ensure the access of workers to shrinkage stopes manways are excavated from the raises.

The broken ore is moved through funnels and short ore-chutes to a scraper subdrift along which it is hauled to the haulageway by a scraper hoist.

Shrinkage Mining in Rooms Along or Across the Strike with Mass-Breakage of Ore by Longholing

General. In the foregoing we discussed mining methods involving the breaking of the ore by blasting short or medium-length holes. This mode of breaking makes it possible to achieve a higher degree of ore extraction near the contacts without contaminating the ore with gangue, but the driller's efficiency is rather low (small yield of ore per 1 metre of blast-hole).

Since drillers work directly under the back of the room its dimensions should ensure full safety for the men.

In working thick and very thick ore bodies with a low content of useful constituents wide use is made of shrinkage stoping with

mass-breakage of the ore. Increased losses and dilution of the ore are in this case offset by higher labour efficiency.

With mass-breakage of the ore the size of the rooms may be appreciably greater than in breaking by means of shallow holes, since the drillers do not work directly under the back of the room.

The successful application of the systems under consideration is closely linked with the introduction of longholing during stoping at a number of mining enterprises of the USSR.

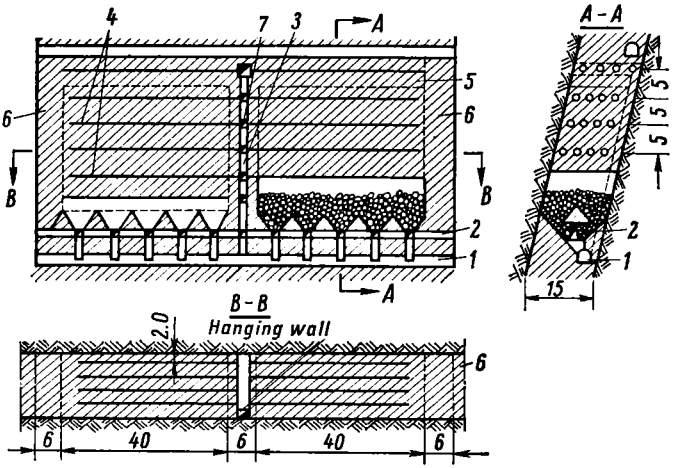


Fig. 185. Shrinkage stoping of ore in rooms along the strike by longholing: *1*—haulage drift; *2*—grizzly drift; *3*—raise; *4*—long holes; *5*—ceiling; *6*—intervening pillar; *7*—drill crossdrift

In this system the ore is extracted in rooms either along or across the strike, depending upon the ore body thickness.

Let us first consider this method with special reference to the extraction of ore in rooms along the strike (Fig. 185). The design characteristics of the system are evident from the drawing. Rounds of parallel blast-holes 35-38 m deep and 88-105 mm in diameter are drilled with heavy drifters from special chambers excavated from drill crossdrifts.

The blast-holes are spaced at 2-3 m in the horizontal and at 4-5 m in the vertical direction, their distance from the contact plane being 1.5-2 m, depending on the blast-hole diameter. Long holes are brought to a point 3-4 m short of the intervening pillar boundary. They are charged with explosives packed in special cartridges up to 1 m long. Stemming, 3-4 m long, consists of a sand-and-clay material; split wooden cylinders are sometimes used for the purpose. Blast-holes are charged by means of screwed-up or extension stem-

mers. Pneumatic charging devices may also be employed. Blast-holes are ignited with the aid of two lines of detonating fuse running

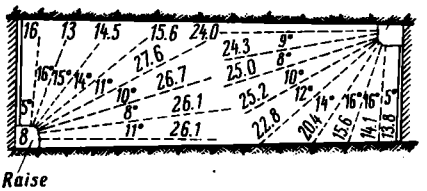


Fig. 186. Breaking of ore by means of fan-shaped rounds of long holes

throughout the length of the blast-hole, along its walls. The charging of the lower row of blast-holes is started only after the undercutting of the room has been completed (when blast-holes in two or three overlying slices have been drilled). The room is undercut so

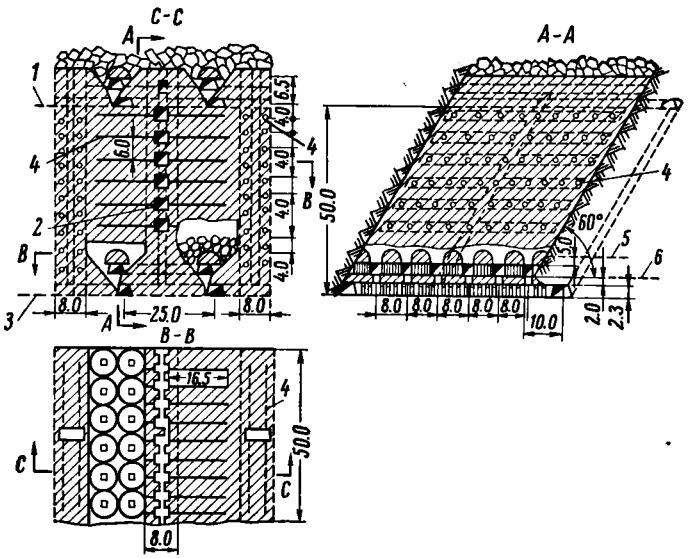


Fig. 187. Shrinkage stoping across the strike with the breakage of the ore by means of parallel deep holes:

1—return air horizon; *2*—drill crossdrifts; *3*—haulage level; *4*—deep holes; *5*—undercutting level; *6*—grizzly level

as to leave the necessary space for the loosened ore after breaking of a slice. With fan-shaped disposition of long holes (Fig. 186) they are drilled from raises driven in the opposite corners of the room. The advantage of this disposition is a lesser amount of subsidiary development work, the drawback being nonuniform fragmentation of the ore and a high yield of oversize, particularly when the ore is hard.

When extracting ore in rooms across the strike it is commonly broken down by means of a round of parallel or fan-shaped long holes drilled from workings driven in intervening pillars. When making parallel blast-holes, it is necessary to run drill crossdrifts, thus increasing the amount of subsidiary development work. The design characteristics of the method of mining by rooms disposed across the strike are represented in Fig. 187, where the basic dimensions of the individual elements of the system are indicated. The methods of mining with mass-breakage of the ore with the aid of chamber charges are not discussed here, since they are rarely used.

Shrinkage Mining with Breast-Stoping of Ore

This method is used in working thin ore bodies. The ore in a level up to 50-60 m high is extracted with a continuous face running on strike or up the raise, its total length reaching 100-130 m. No sill pillar is left, the ore being broken by blasting shallow holes and its subsequent drawing through pull holes. The raises are widely spaced. Communication with stope areas is maintained through ladder compartments of manways made every 20-25 m in the stored ore. The method finds little use.

Basic Advantages and Shortcomings of Shrinkage Stoping Methods

Methods involving the shrinkage mining of ore are highly effective. Their major merits include: small amount of development and subsidiary development work per 1 ton of ore mined; high efficiency of drilling and blasting operations, particularly in variants involving mass-breakage; no need for stope-area support, haulage or overcasting of ore; good aeration of stopes; insignificant losses and dilution in the breaking of ore with the aid of shallow holes; high intensity of mining; considerable stocks of ore in rooms making possible its uniform delivery; comparatively simple design characteristics of the methods; drastic reduction of the dust content in the air and of the scope of drilling operations in breaking ore with the use of long holes; possibility of a wide use of multiface mining and multimachine drilling.

The basic disadvantages of the systems are: strictly defined conditions for their use; impossibility of separating the ore into grades and classes, of its sorting underground (separation of the broken ore into two grades and sorting at the face are possible only in working thin deposits and under a definite set of mining conditions; sorting can also be done in mining lode deposits with special extensible waste-chutes arranged in the stope area, which, however, considerably

complicates the conduct of operations); increased losses and dilution of ore in variants involving mass-breakage, especially in the case of irregular contacts; possible self-ignition and oxidation of sulphide ores after prolonged storage in a shrinkage stope. The technical and economic characteristics of shrinkage stoping methods are listed in Table 21.

Table 21

Technical and Economic Characteristics of Methods Involving Shrinkage Mining of Ore

Mining methods	Coef. of recovery	Dilution factor	Explosives consumption, kg/ton	Output per faceworker per shift, tons
1. Shrinkage stoping of ore in blocks along the strike with ore-breakage by blasting shallow holes	0.93-0.95	0.10-0.15 (with the vein thickness equal to the width of the stope area)	0.50-0.60	8-10
2. Shrinkage stoping in rooms along or across the strike with ore-breakage by blasting shallow holes	0.95-0.97	0.05-0.10	0.40-0.50	15-25
3. Shrinkage stoping in rooms along or across the strike with ore-breakage by longholing	0.90-0.95	0.10-0.15	0.30-0.40	40-50

Note: 1. The characteristics of the variants 2 and 3 are given with special reference to the extraction of ore in rooms. To determine the average figures for the extraction of a block (room and pillars) one should take into account the proportional share of the methods employed in taking out individual portions of the block, and also their characteristics.
2. The dilution factor should be adjusted in each individual instance with due regard for the thickness of the ore body and gangue inclusions. The characteristics of the variant 3 may also be applied to pillaring, which is usually done by efficient methods of breaking.
3. With a radical improvement in the system the characteristics of the variant 1 may be substantially increased (the "Khrustalny" mine practise).

5-4. METHODS OF MINING WITH FILLING OF THE STOPE AREA (CLASS III)

General

This class includes systems in which the sides of the stope area are supported with backfill built up as the working face moves on (only the space directly in front of the face remains temporarily

unpacked). In the filling methods timbering is used as an ancillary or provisional means of support.

These systems of mining are ordinarily employed with ores and host rocks of medium stability in taking ore out from ore bodies ranging from very thin to very thick and occurring at various angles of dip.

The losses and dilution of ore here are insignificant, and with horizontal slicing of ore sorting of the broken muck is possible directly at the working face. The filling methods are safe with regard to fire hazards because little timber is consumed and the worked-out space is packed with incombustible materials.

With tight filling the ground surface and enclosing country rocks can be preserved (not caved), which in many cases may be very important.

Owing to the comparatively high prime cost of mining filling methods are basically employed in working deposits of valuable ores, whose losses and dilution should be kept to a minimum.

The practice of mining lode deposits knows some examples of adopting filling methods in place of shrinkage stoping, since filling makes it possible to substantially cut down the dilution of ore in packing, especially with ore bodies of irregular shape (practice at Canadian mines).

Cut-and-fill methods are used in mining non-ferrous and rare-metal ores with a content of valuable constituents and also in the working of chalcopyrite ores presenting fire hazards.

In recent years the total proportional share of cut-and-fill mining in the USSR has been around 5% of the aggregate production of non-ferrous, rare-metal and gold ores. The proportion of ore tonnages extracted by filling methods from lode deposits is about 20 per cent.

In connection with the wide utilization of machine stowing and the greater depth of mining the relative application of filling methods should increase. Mining with backfilling is often practised in Canadian non-ferrous metal ore deposits. Its share at these mines is about 15% (45 mines). The extensive application of backfilling in Canada has been promoted by the technological achievements in utilizing deslimed mill-tailings as a mine-fill.

Class III includes the following methods employed in working Soviet ore deposits:

1. Block mining along the strike: with bulk extraction of ore in horizontal slices; selective mining of ore and host rock in horizontal slices; with bulk extraction of ore by inclined slicing.

2. Room mining along or across the strike with bulk extraction of ore by horizontal slicing.

Cut-and-Fill Mining in Blocks Along the Strike with Bulk Extraction of Ore in Horizontal Slices

This system may be used with extensible ore-chutes (spaced at 6-8 m) and hand shovelling of ore, or with scraper haulage of ore and fill. Let us consider the variant with scraper haulage as the more effective one.

Typical conditions for the application of this method (Fig. 188), aside from those common to all cut-and-fill systems, are a high dip

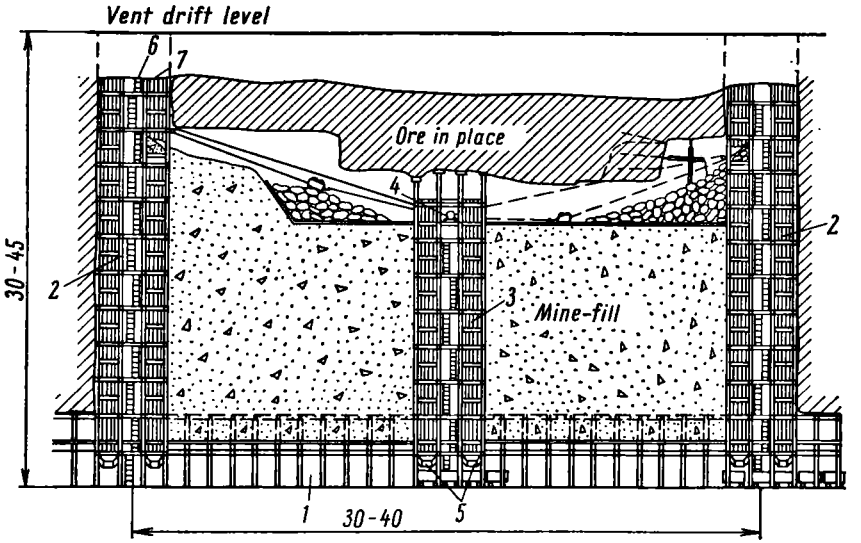


Fig. 188. Cut-and-fill mining with horizontal slicing and stowing in blocks on strike with scraper haulage of ore and fill:
1—haulage drift; *2*—raise; *3*—extensible ore-chute; *4*—scraper hoist; *5*—draw hole; *6*—ladderway; *7*—waste-chute

and a limited thickness of ore bodies (predominantly very thin and thin ones). In this method a horizon 30-45 m high is broken up into separate blocks up to 40-50 m long. To prepare them for stoping two or three compartment raises and drifts are excavated. No subsidiary development workings are driven. Stoping operations are usually started from the back of a supported drift without leaving any sill pillar.

The first slice is taken starting from a raise. To avoid drift timber breakage while extracting the first slice ore is broken by shooting shallow holes with small explosive charges. The broken ore is drawn onto the roadway through pull-holes in the lagging. After the first

slice has been taken out a strong flooring made up of round lagging or thick boards is laid down on timber pieces of the drift and an ore chute is made in the centre of the block. As the second slice is being extracted the stoped-out space of the first slice is packed with mine-fill and ore-passes are built up over the chute. The ore-pass is supported with closely set crib or stull timbering, boarded inside and outside.

Ore in a slice is broken by shooting horizontal or vertical holes. These are drilled with drifters or stopers. The broken ore is hauled to an ore-pass and dropped into it. A wooden board or steel sheet

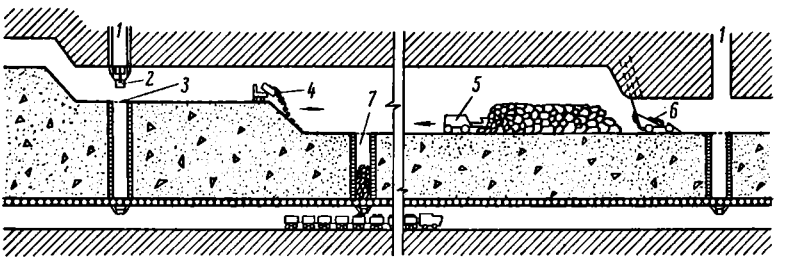


Fig. 189. Horizontal slicing-and-filling with the use of self-propelled equipment: *1*—mine-fill raise; *2*—pull-chute for mine-fill; *3*—grizzly; *4*—self-propelled mine-car for haulage of fill; *5*—self-discharging and sell-loading hauling unit; *6*—drill carriage; *7*—ore-chute

flooring should be laid down on the fill prior to the breaking of ore. With a particularly valuable ore the flooring is laid down on a spread tarpauline to prevent the spilling of fine ore through the flooring, which is then taken off before stowing operations are started.

Timely laying down of flooring is a major prerequisite for minimizing ore losses into the fill. Failing this, metal losses in the ore may rise from 3-5% to 10-15% and more, especially when the ore is comminution-prone, with shifting of the useful component to the fine fractions. The packing material is supplied via a raise and brought to the site of final disposal with a scraper (Fig. 188).

In order to combine drilling with mucking and sorting of the broken ore the holes at the working face can be drilled by standing on ore piles. This greatly increases the intensity of stoping. A disadvantage of such synchronization is the exposure of the enclosing country rocks to a height of two slices, which is not always advisable in view of the possible insufficient stability of the ground.

The ore in a block is taken out up to the ceiling. The top slice is backfilled by shovelling.

Among the measures to raise the efficiency of drilling and the haulage of ore and mine-fill in this system mention should be made

of the use of small self-propelled drill carriages, self-loading hauling units for handling ore, and self-propelled mine-cars for the mine-fill. Drill carriages operate in slices with increased headroom, they are easy to manipulate and can service a number of blocks simultaneously. This mobile equipment is successfully employed at the

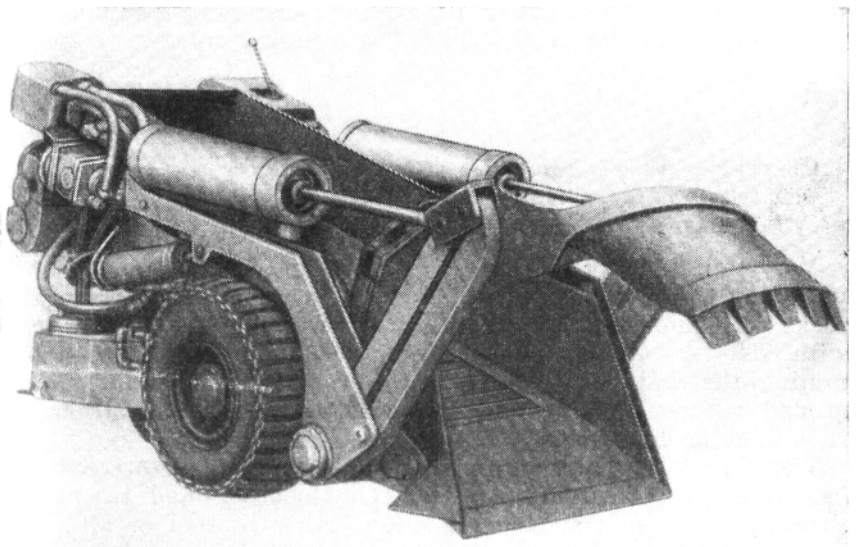


Fig. 190. ПДВ-2 machine

Italian mine "Monteveccio" (Fig. 189). Soviet-manufactured loading and hauling units of the ПДВ-2 type (Fig. 190) may find wide application in this method. The rated capacity of this machine for loading and haulage to a distance of 50 m is about 10 m^3/hr; it is powered by compressed air and is small in size (width 1,280 mm, dumping height 1,800 mm). The unit weighs 3,800 kg and has a minimum swinging radius of 2 m.

A measure that can substantially reduce metal losses with the system under consideration is gunniting the surface of the pack bulk. Prior to breaking the ore in a slice the surface of the fill mass is thoroughly levelled out and covered with a quick-setting concrete grout 15-20 cm thick by using a spray-gun. The practice of Canadian nickel mines shows that with floorings made of Portland cement and deslimed tailings taken at a ratio of 1:7 it acquires within 3 days a strength permitting mobile drills and combination loading-and-transport equipment to travel on the flooring, the ore being

broken down onto it. The quickness with which such a covering can be built and its compactness, along with its low cost have promoted the wide application of gunniting in working rich ores by cut-and-fill methods (practice adopted at the mines in the USA, Canada, the GFR and other countries). One of the basic requirements for gunniting is the mechanization of the placement of concrete with broad utilization of mobile units moving concrete by means of compressed air at a pressure of 7 atm along a supply pipeline 150 mm in diameter. The capacity of one such unit is up to 15 cu m of concrete mix per shift.

Cut-and-Fill Mining in Blocks Along the Strike with Selective Stoping of Ore and Host Rocks by Horizontal Slicing

Very thin veins or thread veinlets of highly valuable ore can be mined by two methods: with bulk breaking of ore and part of host rocks throughout the entire width of the working face (0.8-1.0 m) or with selective stoping of the ore and enclosing rocks. With bulk mining, the waste is sorted out in the stope and additionally on the ground surface. Should underground sorting involve any difficulties it is done on the surface.

The advantage of bulk mining in comparison with selective stoping is a lower cost of actual extraction. On the other hand, the possibilities of subsequent sorting are often rather limited or altogether non-existent, since the ore cannot be separated from waste by the colour, especially when they have become mixed together.

When sorting is impossible after waste gets into the ore during bulk mining the cost of transporting muck to a mill rises and the expenditures on dressing may also increase, while the extraction of metal reduces, as in the mining of tungsten and other rare-metal ores. Selective stoping permits dilution to be cut down by 2-4 times.

In such cases it is necessary to compare the technical and economic characteristics of selective and bulk extraction of the vein matter and host rocks with due allowance for transportation and the processing technology at the concentration mill.

The basic conditions for selective extraction of ore and enclosing country rocks are: clearly defined boundaries of the vein and the presence of a selvage, at least on one of its sides; no mineralization of wall rocks, for in selective mining the waste is used as packing and is left in the worked-out space; regular occurrence, steep dip and vein thickness below 0.5-0.4 m, since with a greater thickness selective extraction is usually inexpedient. An ore body is ordinarily taken out in levels 30-40 m high. A larger level interval would impede detailed exploration of very thin ore bodies.

The general sequence of preparation of a block for stoping remains unchanged. The on-the-spot availability of the fill (broken barren rock) makes it possible to increase the length of a block from 30 to 60-80 m, with setting up of an additional ladderway compartment in one of the ore-chutes in the middle of the block. The muck in the block can be broken with one or several overhand stopes. Use is also made of separate extraction of gangue and vein matter over the

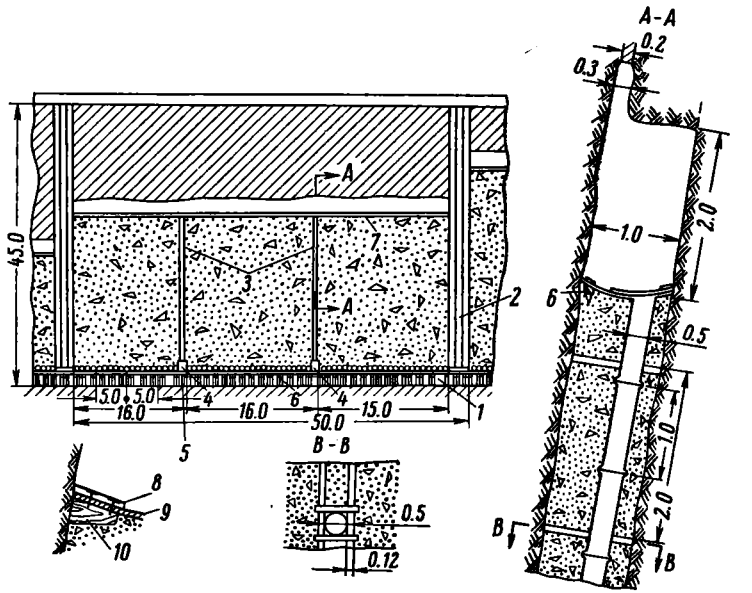


Fig. 191. Variant of selective mining of ore and host rocks over the whole area of a slice:

1—haulage drift; *2*—raise; *3*—ore dump pipes; *4*—room; *5*—chutes; *6*—round lagging; *7*—flooring; *8*—board; *9*—conveyer belt flooring; *10*—wedge

entire horizontal area of the layer with the vein matter and the excess waste drawn via steel dump pipes in place of costly ore-passes which take a long time to build. This practice ensures ample face room, a strong horizontal flooring and, at the same time, the use of a large number of air hammers (Fig. 191).

The merits of the variant under review are: high output per driller (from 5-6 up to 14-15 cu m per shift), high labour productivity per faceworker ranging from 1.5 to 3-4 cu m of muck per shift, depending upon the width of the stope area and the hardness of the ground; low timber consumption (0.01 cu m per 1 cu m of muck); a considerable reduction in losses and dilution due to the use of horizontal floorings made of steel sheets or rubber strips; high inten-

sity of stoping operations, reaching 8-12 m/month up or down the block; the possibility of subsequent drawing of waste if it contains useful components.

In selective stoping barren rock can be broken down first, the vein matter coming next; the sequence of extraction may, however, be reversed when the vein matter is softer than the enclosing country rocks. The sequence of breaking is also influenced by the thickness of the vein.

To avoid losses of vein matter the holes for the blasting of the enclosing rocks are located so as to prevent caving of the ore, leaving a layer of barren ground 10-15 cm thick along the contact with the vein matter. For more complete extraction of ore with an insufficiently distinct contact a narrow layer of barren rock is broken together with the ore on the opposite side. In the presence of mineralization in stable host rocks and when it is advisable to separate the rich vein matter from the lean ore represented by mineralized enclosing rocks one may first break the vein matter down onto the flooring and then move it to ore-chutes, with the mineralized rock extracted afterwards (after the flooring is removed), stored for a time and then pulled out through draw-holes made in the back of the haulageway.

Vein matter of low hardness is mined out by using air hammers. These enable the vein matter to be broken away strictly along the contact surfaces with very slight losses and dilution. When the waste obtained in the stope is used as a fill the deficiency or excess in the thickness of the gangue layer can be estimated proceeding from the thickness of the vein and the thickness of the gangue layer required to fill the working space. The thickness of a rock layer, M_r, required to ensure the filling of the working space can be determined from the formula

$$M_r = \frac{M_w \cdot K_1}{K_2}$$

where M_w = the adopted width of the working area, m;
K_1 = coefficient of filling of the stoped-out area (taking into account the presence of ore-chutes);
K_2 = coefficient of loosening of the ground after breakage with allowance for its consolidation.

Example: Using the above formula and taking $M_w = 0.8$ m, $K_1 = 0.8$ and $K_2 = 1.5$, we find $M_r = 0.43$ m.

Accordingly, with selective extraction of a vein 0.3 m wide the excessive thickness of the barren rock layer is 0.07 m and in this case the waste should be removed from the stope area. With selective extraction of a vein 0.5 m thick the deficiency in the thickness of the rock layer is 0.13 m and therefore additional fill is to be furnished from outside.

With this method ventilation presents no difficulty and is realized by diverting the air current from a haulageway to a vent drift via block raises and worked-out slices.

The technical and economic characteristics of stoping are listed in Table 22, p. 350.

Cut-and-Fill Mining of Thin, Gently Sloping Deposits

Cut-and-fill mining by strips to the rise (Fig. 192). This system is employed in working ore bodies of small size with an unstable back. A block is divided into a series of strips worked out to the rise.

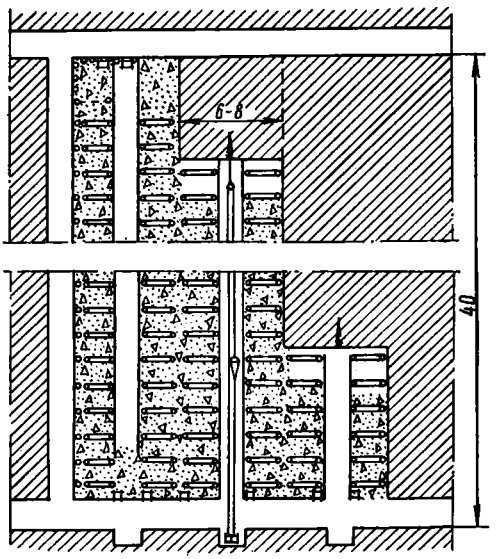


Fig. 192. Mining by strips to the rise with timbering and filling

Strips 6-8 m wide are taken out with an advance of 4-10 m. Ore and rock are extracted separately or together by firing shallow holes 1.2-1.5 m deep. With joint breakage the gangue is picked to be subsequently used for packing the mined-out space. In unstable ground the roof of a stope face is supported with props or frame sets and the broken ore is hauled to the drift with a scraper, or made to slide down along a trough. The rate of advance of a stope face is 6-10 m per month.

Breast-stoping with selective mining and subsequent backfilling (Fig. 193). This system is exemplified by the mode of extraction practised in the GDR in working a bedded deposit of copper slates

15-40 cm thick. The ore is taken out up the rise with a long continuous stope divided into two parts. The ore is first broken off along a narrow slot, then the roof is blasted. The total height of the stope area is 0.8 m. The ore and barren rock are blasted out by shooting small-diameter (24-30 mm) holes. Drilling is done with light automatic-feed jackhammers, which are mounted on light anchorable mobile

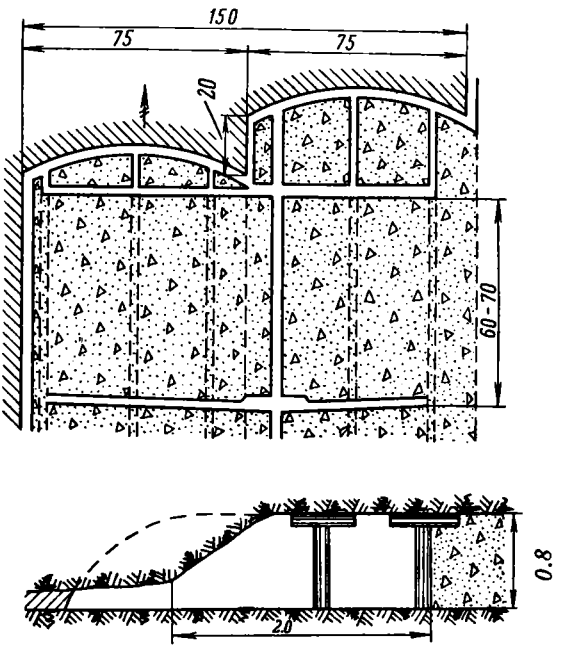


Fig. 193. Up-raise breast-stoping with selective mining

drill carriages. The drilling efficiency per jackhammer per shift is 60-80 holes, 1.2-1.5 m deep. The ore is brought to the slide chute by mobile plate conveyers or in special monorail mine-cars.

Cut-and-Fill Mining in Blocks Along the Strike with Bulk Extraction of Ore by Inclined Slicing

This method (Fig. 194) was formerly widely used in the copper mines of the Urals. Its specific feature is that the stoping of ore and the disposal of the fill are done in slices running at an angle of 30-40° to the horizontal. In inclined slicing the mine-fill can be moved

along the stope area and the ore brought down to ore chutes by gravity, which reduces the cost of stoping.

The conditions typical of this system (apart from those common to all cut-and-fill methods) are ore-body thickness from 0.8 to 3-4 m; dip angle of at least 60°; no need to sort the ore at the stope face.

The ore body is worked out in horizons 35-45 m high. With a greater height the horizon is divided into sublevels. Mining with smaller

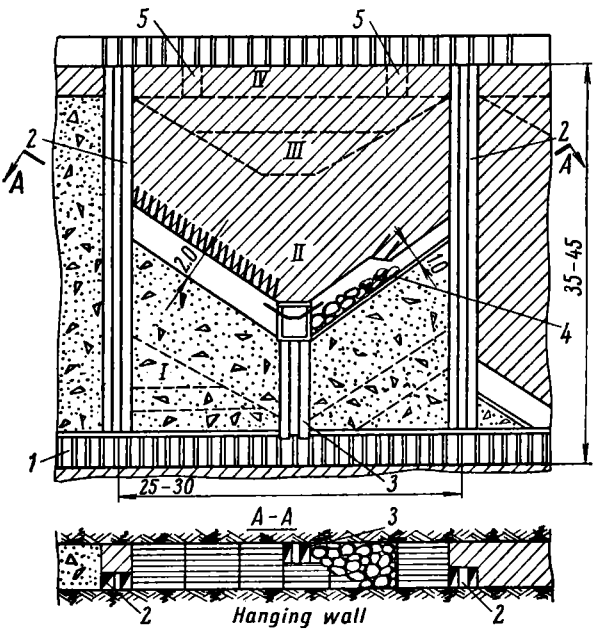


Fig. 194. Inclined slicing-and-filling

level intervals is inexpedient, since the advantages offered by inclined slicing would not be adequately utilized.

A level is divided into blocks 25-30 m long each. The length of a block should correspond to the permissible areas of roof exposure. A block can be extracted by one- or two-way mining (on one or both sides). One-way mining is practised in ores with pronounced jointing, where extraction is only possible in one direction, normal to the jointing. This point should be kept in mind to avoid intensive exfoliation of ore that may result in accidents. In this system a block is developed by driving a reef drift (*1*) and a raise (*2*). When development work is over the bottom portion (*I*) of the block is extracted either in horizontal or in inclined slices, following which

an ore-chute is made in the middle of the block. The major part (*II*) of the block is taken out by inclined slicing with the ore drawn through the central ore-chute (*3*) along a flooring (*4*). The top portion (*III*) of the block is worked out up to the ceiling by horizontal slicing with the fill supplied via short waste-chutes (*5*).

The field of application of this system is restricted by the severe conditions and the difficulties attending extraction of the bottom and, especially, the top portion of the block. Among the promising

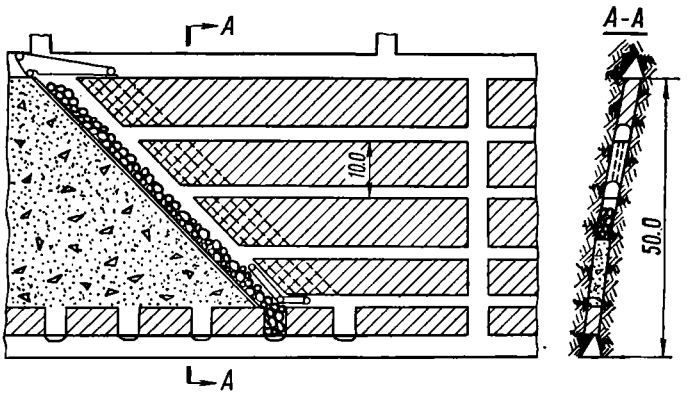


Fig. 195. Inclined slicing from sublevel openings with subsequent stowing and breaking of ore by longholing in Canadian mines

variants of the method is the version involving breakage of ore by longholing from sublevel openings with partial scraping of the ore and setting up a flooring made of used steel ropes and running along the entire length of the inclined slice (practice adopted at Canadian gold mines) (Fig. 195). The output per faceworker reaches 6-8 cu m (15-20 tons) per shift.

Cut-and-Fill Methods of Room Mining Along or Across the Strike with Ore Extracted by Horizontal Slicing

Apart from the conditions common to the use of methods with filling of the worked-out space, the typical conditions for the application of these systems are: ore-body thickness exceeding 6 m; dip angle of at least 50-55°; in working very thick deposits this angle may be less than 50°.

When mining rooms across the strike the stability of the host rocks may be somewhat lower than usual, since they become exposed over small areas.

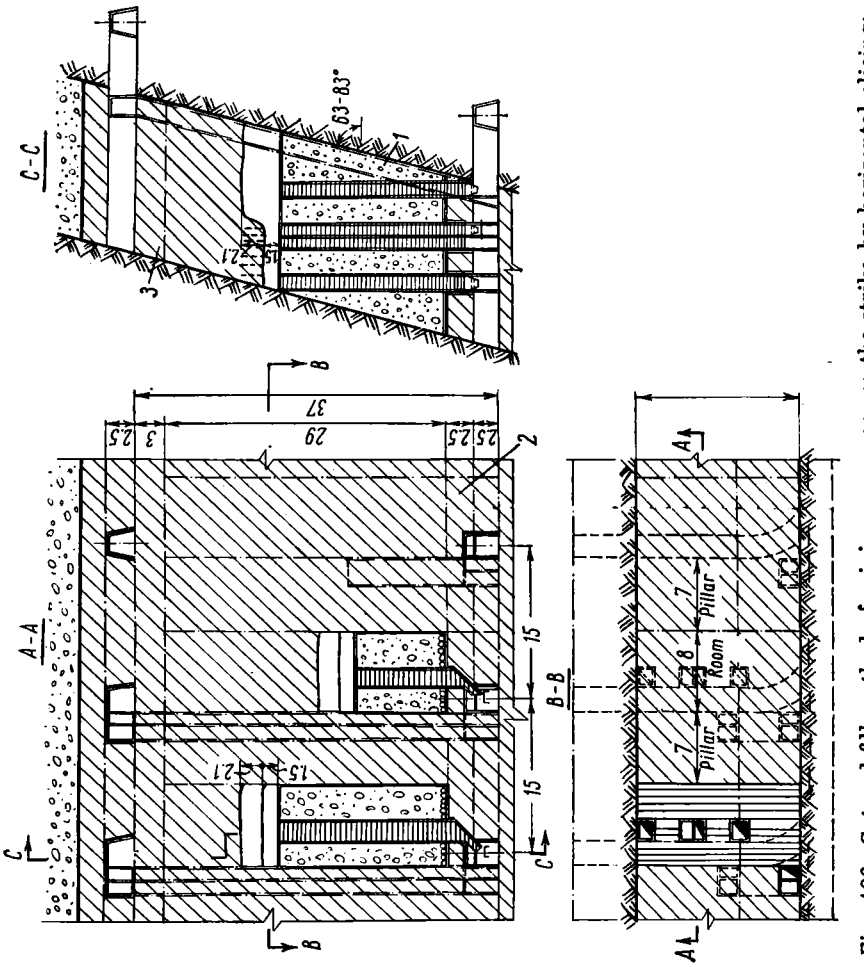


Fig. 196. Cut-and-fill method of mining rooms across the strike by horizontal slicing:
1 — raise; *2* — sill pillar; *3* — ceiling

The method allows the ore to be sorted directly at the working face. The ore body is worked out by levels 30-45 m high; with a greater interval the horizon is divided into sublevels. In ore bodies up to 10-12 m thick stoping is done in rooms along the strike, with greater thickness in rooms across the strike. The room length is chosen so that with a stoping width from 6 to 10 m, depending upon the stability of ore, the exposed area of the back does not exceed the permissible limits of 150-200 sq m. With stoping on strike the width of intervening pillars is at least 8-10 m, whereas in mining across the strike it usually equals that of the room, but should not be below 6 m.

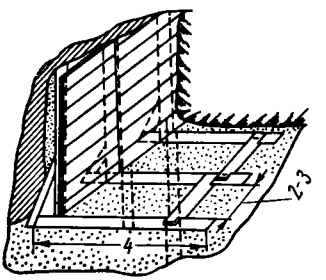


Fig. 197. Protective wall

Intervening pillars are recovered by the pillar-and-stall or vertical-slicing method, either with square set support and filling (when caving of host rocks is not allowed), or by top-slicing (see Classes V to VI of the mining methods). Here we shall give a brief description of the method involving the stoping of ore in rooms across the strike.

Development workings in this system (Fig. 196) are: a reef or, more often, a lateral (stone) drift made in the foot wall of the deposit; crosscuts running from the drift along the boundary line between rooms and pillars; raises near the hanging or foot wall on the boundary line between pillars and rooms. With long rooms an additional raise is put up approximately in the centre of the room.

Crossdrifts and raises are used for the extraction of ore in rooms and for the robbing of pillars.

Stoping in the room is done similarly to cut-and-fill mining by blocks along the strike. Triple-drum scraper hoists with remote control can be used extensively for the hauling of ore and packing material. The number of ore-chutes in a room may, in this case, be reduced to one.

To preclude losses and dilution of ore during subsequent recovery of pillars wooden protective walls should be put up on the boundary with intervening pillars (Fig. 197). Such a wall consists of posts 3 m long set up on the pillar boundary every 1.5-2 m, and board lagging.

The posts are fastened to horizontal sills laid down on the fill (tenons on the butt ends of the posts are inserted into slots cut in the sills) and then boarded. When self-cementing fill is used there is no need for protective walls.

In all systems of mining using mine-fill the multimachine method of drilling operations can be successfully applied with several stopers simultaneously covering a considerable area of the ore stope back, along with mechanized haulage of ore and stowing materials, and also hydraulic filling as described above.

Basic Advantages and Shortcomings of Cut-and-Fill Methods of Mining

The merits include: high recovery of ore, provided losses of ore fines in the fill are eliminated; low dilution of ore; the possibility of underground sorting and grading in horizontal slicing; tight filling makes it possible to preserve the ground surface and the solid mass of enclosing country rocks without substantial changes; the system is fire-safe; the possibility of simultaneous extraction of several levels, which ensures high mining intensity; the possibility of application of multiface and multimachine methods.

The disadvantages include: high labour expenditure in filling operations; difficulty of sorting the ore when it is taken out by inclined slices; complex and inefficient extraction of the top portion of blocks in inclined slicing; relatively high net cost of ore.

The technical and economic characteristics of the cut-and-fill methods are presented in Table 22.

5-5. METHODS OF MINING WITH ARTIFICIAL SUPPORT OF THE STOPE AREA (CLASS IV)

General

As distinct from the open-stope systems of mining, where timbering serves auxiliary purposes, mining with artificially supported stopes is characterized by regular erection of timbering, which is the basic means of maintaining the stope area.

Timbered stopes are used in working deposits of any shape dipping at various angles, preferably in comparatively thin ore bodies (not over 4-5 m thick). In ore bodies of great thickness timbering of the stope area is a very complex and hazardous matter, especially at sites exposed to increased rock pressure. In such cases timbering is supplemented with mine-fill and the method should then be referred to Class V.

Table 22

Technical and Economic Characteristics of Cut-and-Fill Methods of Mining

Mining systems	Coef. of recovery	Dilution factor	Timber consumption, m^3/ton	Explosives consumption, kg/ton	Output per faceworker per shift, tons
1. Block mining on strike with bulk extraction by horizontal slicing	0.95-0.97	0.05-0.07 (in an ore body not less than 0.8-1 m thick)	0.01-0.015	0.2-0.3	4-5
2. Block mining along the strike with selective extraction of ore and host rocks (per 1 ton of vein matter with vein thickness of 0.2-0.3 m)	0.93-0.95	0.15-0.30 (with regular thickness and distinct contacts)	0.03-0.04	0.6-0.9	1.5-3
3. Block mining on strike with bulk extraction of ore by inclined slicing	0.95-0.97	0.07-0.10 (in an ore body not less than 0.8-1.0 m thick)	0.01-0.015	0.2-0.3	5-7
4. Room mining along or across the strike with ore extracted by horizontal slicing	0.95-0.97	0.03-0.05	0.005-0.01	0.2-0.3	10-12

Note: 1. Figures for system 4 are given with special reference to the extraction of ore in a room; to find the mean figures for rooms and pillars allowance should be made for the proportional use of the corresponding mining systems and their characteristics. In room mining on strike (small proportion of pillaring) the data on room extraction may be extended to the entire block.
2. In comparing the effectiveness of methods 2 and 1 due consideration should be given to the transportation costs and the data on ore dressing at the concentration plant.
3. The dilution factors for methods 1, 2 and 3 should be adjusted in each particular case in relation to the thickness of the ore body to be mined, gangue inclusions and sharpness of contacts.
4. With the use of steel pipes for the dumping of ore timber consumption in system 2 can be cut down considerably.
5. With ore-breakage by shooting blast-holes, mechanized haulage of ore and mine-fill and the use of improved flooring labour efficiency can be increased (up to 8-10 tons with system 1 and up to 15 tons and more with system 4). The utilization of mobile equipment and self-cementing fillis of prime importance in raising the efficiency of the cut-and-fill methods.

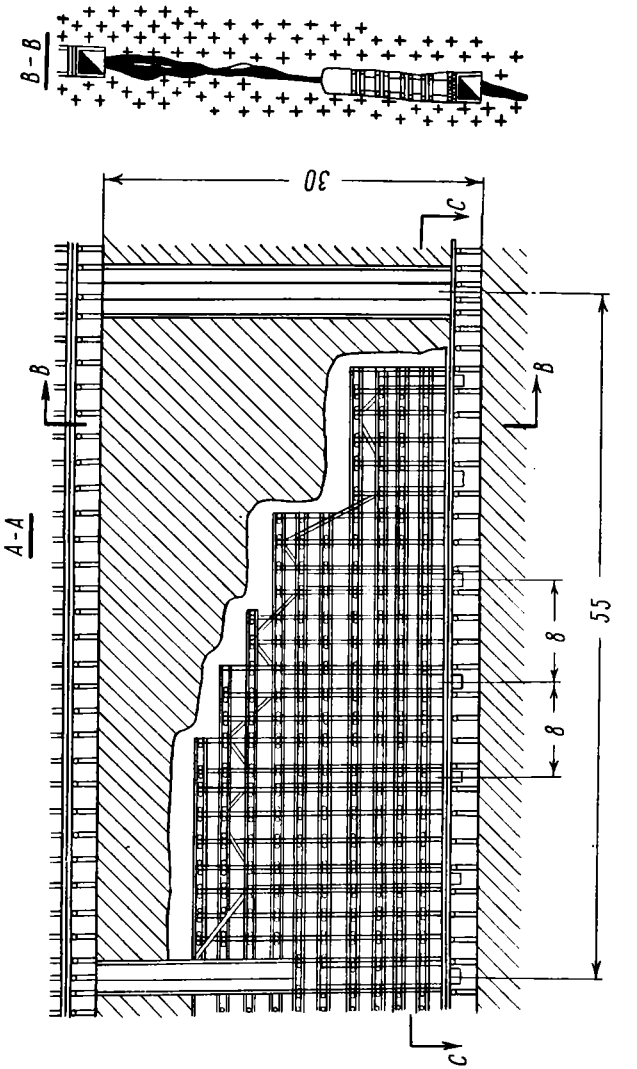


Fig. 198. Stull and square-set mining with ore stoped in blocks on strike (section of half-block)

In the USSR the proportional share of methods using timbering in the working of gold and non-ferrous metal ore deposits is rather small.

Class IV includes the following systems applied in the mining of Soviet ore deposits: block mining on strike with heavy stull timbering, square-set and closed-frame stoping.

Block Mining on Strike with Heavy Stull Timbering

As regards its design characteristics this system differs little from the above-discussed system with stull timbering (see Class I of the methods), its distinctive feature being a regular and closer spacing of stull props, which are not removed and remain in the stoped-out space until the block is worked out completely. With this method draw-holes for the dumping of ore are commonly made by lagging the stull props. The ore broken at the faces slides down to the draw-holes along inclined troughs.

Block Mining on Strike with Stull and Square-Set Timbering

Below we give a brief description of the method as applied at one of the Soviet gold mines (Fig. 198).

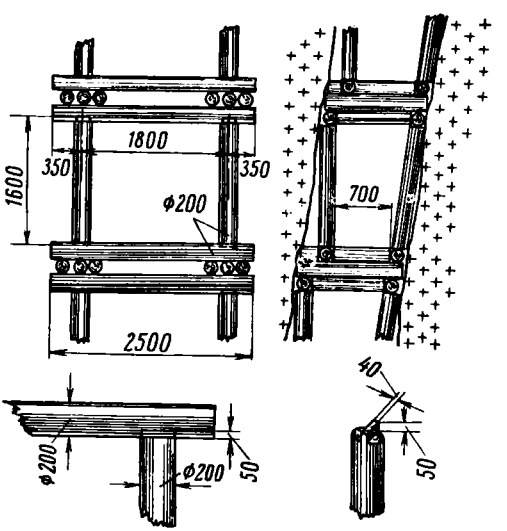


Fig. 199. Details of stull and square-set timbering

The conditions attending the exploitation of the deposit are as follows: it represents a quartz-reef with a mean thickness of 0.35 m dipping at 85°, ore hardness $f = 18$. The hanging wall consists of heavily fractured, low-stability modified granites liable to exfoliation and inrushes, rock hardness $f = 13$-15. The foot wall consists of slightly fractured, low-stability granites, which, when undermined, are liable to exfoliation and inrushes, rock hardness $f = 13$-15. In this method a horizon is divided into blocks up to 100-110 m long each. The development openings are drifts and raises driven along the boundaries and in the centre of a block. The ore in blocks is extracted by overhand stopes with simultaneous sorting out of gangue, which is thrust down into the space between the ore-chutes.

As the stope faces move on, the worked-out space is immediately supported with reinforced timbering—square-sets made up of stull props, sills and upright strain posts (Fig. 199). To facilitate the dumping of the broken ore into draw-holes wooden troughs fixed to strain posts are used.

The extraction of blocks in gently sloping ore bodies 0.8-1.0 m thick with supported stopes is usually done in strips up the rise, use being made of frame sets and randomly arranged cribs. The top and bottom drifts are fenced off from the stoped-out space by ore pillars, leaving passages for men and operating scrapers. The length and height of a block vary between 40 and 50 m.

Advantages and Drawbacks of Methods with Artificial Support of the Stope Area

The main advantages of these systems are: a high coefficient of recovery and low dilution of ore (this merit is particularly obvious in the working of intricately shaped deposits with frequent changes in the mode of occurrence); the possibility of underground sorting of the broken muck and of leaving the barren rock in the mined-out space; flexibility, i.e. the possibility of making changes in individual elements of the system, in the design of the timbering and in the modes of breakage.

The principal shortcomings of the systems are: high consumption of mine timber, which reaches 8 to 15% of the stoped-out space in the case of reinforced stulls, 5 to 12% with square-sets, and 15 to 22% with heavy frame-set timbering; low output per faceworker because much of the working time is spent on setting up timber; high fire hazard; limitations imposed on drilling and blasting by the mandatory use of shallow holes and small charges to avoid the knocking down of timber pieces.

Owing to the substantial demerits of the methods of mining with timbered stopes they find little use.

Technical and Economic Characteristics of Methods with Timbered Stopes in Working Thin and Very Thin Ore Bodies

Coefficient of recovery 0.95-0.97. The dilution factor may be taken as 0.05-0.10 depending upon the thickness of the lode and the width of the stope area (with an ore body not less than 0.8-1.0 m thick). Timber consumption 0.02-0.07 cu m per 1 ton of ore. Explosives consumption 0.35-0.40 kg per 1 ton of ore. Output per face-man 5-6 tons per shift.

5-6. METHODS OF MINING WITH ARTIFICIALLY SUPPORTED AND FILLED STOPE AREA (CLASS V)

General

Such systems are employed with soft and unstable ores and host rocks, when working ore bodies of different thickness dipping at different angles. They are among the methods characterized by the highest cost of mining. On the other hand, they involve only slight losses and dilution of ore. These methods allow easy sorting and delivery of different grades of ore, which is of prime importance in extraction of very valuable ores or ores requiring complex processing technology (non-ferrous and rare metals). Methods of mining with a timbered and filled stope area, like those with compact packing, enable the ground surface and enclosing rocks to be protected from subsidence and caving.

At Soviet mines the proportional share of the methods discussed in the total production of ores has been insignificant in recent years, accounting for only 2% of the non-ferrous ore output. At gold mines their share is somewhat greater.

The relatively little share of these systems in the working of non-ferrous metal mines is attributed to the wide-scale application of top-slicing under similar conditions, except where the ground surface and enclosing country rocks have to be protected against caving. The greater proportional use of mining with artificially supported and filled stope areas at gold mines is due to the limited thickness of ore bodies which impedes the application of top-slicing methods.

The methods with artificially supported and filled stopes are not employed in mining iron ores.

Class V includes the following systems used in working Soviet ore deposits:

1. Block mining methods: in blocks along the strike with frame-set support and filling; in short blocks with square-set stoping-

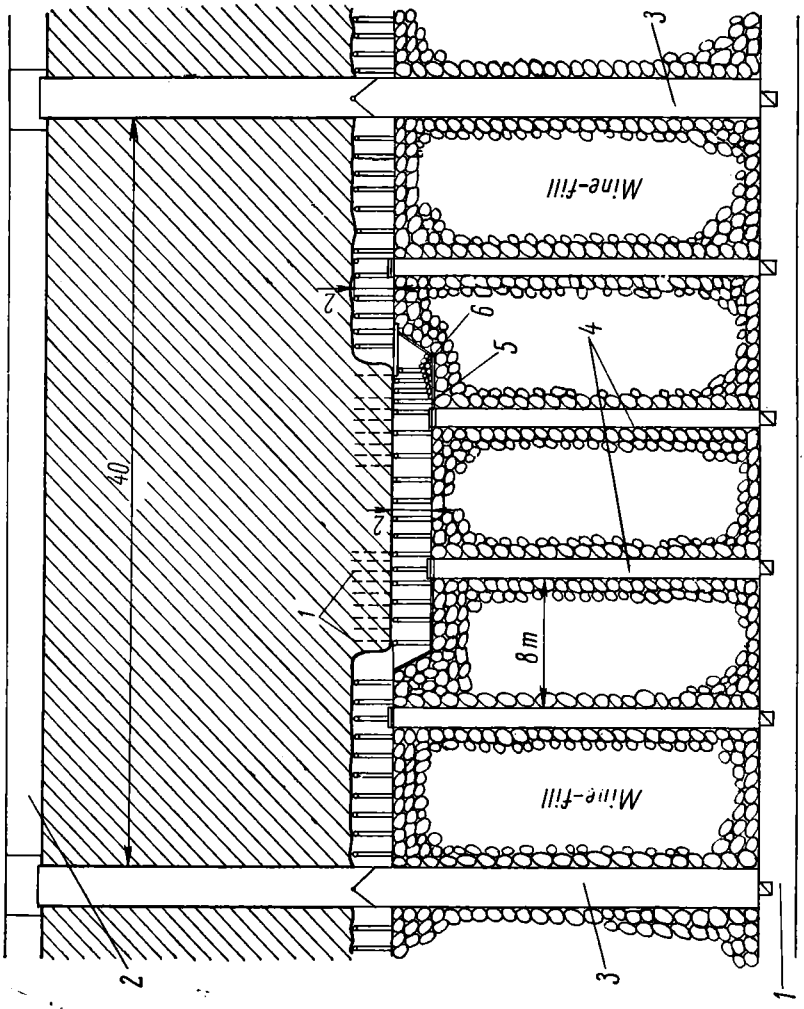


Fig. 200. Variant of block mining on strike with frame-set support, filling and overhand stoping:
1—haulage drift; *2*—airway; *3*—raise; *4*—ore-chute; *5*—flooring; *6*—broken ore; *7*—blast-holes

plus-filling; by vertical slicing (in narrow blocks) with square-set stoping-plus-filling; in blocks with downward stoping, artificial support and filling.

2. Methods of mining in rooms along or across the strike with square-set stoping-plus-filling.

3. Methods of mining with breast-stoping of ore, square-set support and filling.

Mining in Blocks Along the Strike with Frame-Set Support and Filling

The main variant of this method involves horizontal slicing with overhand stoping and drawing of ore through ore-chutes (Fig. 200). This variant is discussed below.

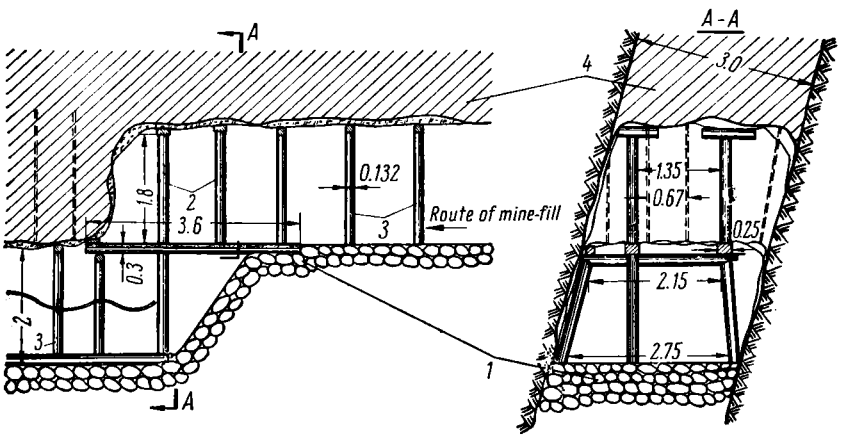


Fig. 201. Details of temporary timbering put up in a stope in an ore body up to 3 m thick:

1—mine-fill; *2*—temporary timbering; *3*—permanent timbering; *4*—solid mass of ore in place

The typical conditions for the use of the system are: very unstable ore and wall rocks; small thickness of the ore body (maximum 2.5-3 m); a dip angle exceeding 45°; valuable ore.

With this method mining is done in levels with a height interval of 30-40 m. Each level is divided into blocks 40-60 m long which are worked either in advance or retreat order. Stoping can be done concurrently in several levels.

Development workings in a block are a haulage and a vent drifts run in the ore and two- or three-compartment raises situated at the block boundary. Drifts are supported with frame-sets, their

roof and walls being lagged with slabs. Raises are supported with closely set cribs.

Stoping begins with the extraction of the first slice 2 m high, directly above the back of the drift, or leaving a sill pillar 2-3 m high. This is usually abandoned when the level is worked in advance order and is recovered during the second mining of the level.

The ore in a block is taken out in horizontal slices, each with two overhand stopes. In a stope the ore is broken by shooting up-holes, the stoped-out space being immediately supported with temporary timbering (Fig. 201). As the stope face advances permanent timbering (three-piece frame-sets) are put up. The broken ore is brought to ore-passes extended in the block every 8-10 m.

Packing material is stowed by dumping. Its delivery from the raise to the site of disposal may be done in rocker-type mine-cars, or with scrapers.

Under certain conditions frame-sets may be partly replaced by rock bolting, which considerably reduces the cost of timbering. Timber support may in part be replaced by metal props, ordinary floorings by concrete ones, with self-cementing fill passed from the ground surface down along pipes.

Mining in Short Blocks with Square-Set Stoping and Filling

This system (Fig. 202) can be employed independently in extracting individual ore bodies, or as an auxiliary method for recovery of intervening and floor pillars abandoned when using other methods of mining. The system is described below with special reference to the working of an entire ore body.

In addition to the common conditions noted earlier the conditions typical of this system are: very unstable ore and wall rocks, considerable thickness of ore bodies, different angles of dip.

An ore body is worked in levels not exceeding 30 m in height. With a greater height interval the level is divided into sublevels. Each sublevel is broken up into short blocks whose size is multiple of the number of square-sets, usually 4×6 or 4×8.

The extraction of short blocks is commonly done in the direction from the hanging wall towards the foot wall and from the centre to the flanks of the ore body (Fig. 203). Each block is divided into sections (in Fig. 202 into 4 sections). The advance rate of one section with regard to the other should be at least four rows of square-sets. During extraction each block borders on the solid mass of ore in place and on the gobbed space of the neighbouring blocks.

The small area of the section to be worked out ensures high mining intensity and extraction without exfoliation of the roof. Experience

in the mining of ore in short blocks and by vertical slicing points to the possibility of safe operation under the most trying conditions.

Development workings in this system are: a lateral drift in the foot wall, with a reef drift being sometimes driven in the hanging wall for ancillary purposes; crosscuts driven near the raises in each row of short blocks; two-compartment raises put up in the centre of each short block. The raises ensure the passage of men, the dumping of broken ore onto the haulage floor, the delivery of mine-fill and timber, and the flitting of the necessary equipment from the upper horizon down to the working faces.

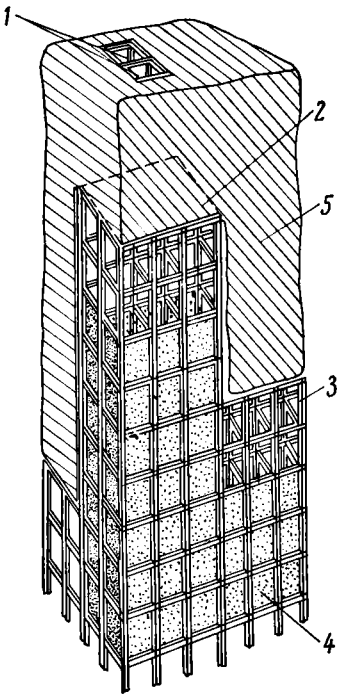


Fig. 202. Stoping in a short ore block:
1—raise; 2—block section; 3—square-set; 4—mine-fill; 5—solid ore in place

The ore in sections is broken starting from raises, its extraction being done in "cubes", with the stoped-out area immediately supported with square-sets (Fig. 202).

In this and other square-set stoping methods the size of square-sets is as follows: height from 1.8 to 2.4 m; dimensions in plan from 1.25 × 1.25 to 2 × 2 m, commonly 1.5 × 1.5 and 1.8 × 1.8 m (square-sets of larger size are used with the more stable ores).

Square-sets are usually made of round timbers 200-400 mm thick and rarely of square timbers.

The following rules must be strictly observed in square-set stoping: the props should be installed vertically, the caps and girths laid down horizontally, with the square-sets thoroughly wedged.

The design features of timbering have been described above.

One or two slices are worked in a section, depending upon the stability of the ore. The fill is either shovelled up or dumped. Within a section the ore and packing material are, as a rule, handled by hand because of the short hauling distances. Along the boundaries of the extraction area square-sets are lagged with boards or slabs, and prior to ore breakage a flooring is set up in the slice to preclude

contamination of the ore with waste. Strict adherence to the stipulated sequence of working short blocks and sections, and rapid extraction of individual slices are conditions of prime importance.

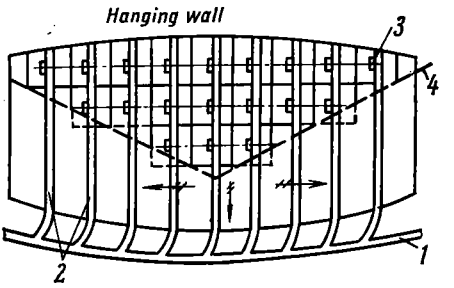


Fig. 203. Haulage level plan and the sequence of extraction of short blocks: *1*—lateral drift; *2*—crossdrifts; *3*—two-compartment raise; *4*—block-extraction line; arrows indicate general direction of stoping

Ventilation in this system presents no difficulty, since the air current from the lower horizon flows along the raises up to the upper level, sweeping the working faces on its way.

Vertical Slicing (in Narrow Blocks) with Square-Set Stoping Plus Filling

The conditions for the application of this system (Fig. 204) are the same as for short block mining.

The ore body is taken in levels maximum 30 m high. When the level interval exceeds 30 m it is divided into sublevels. Each level is broken up into individual slices two or three square-sets wide extending across the entire thickness of the deposit. The slices are worked out from the centre towards the flanks. Each slice is extracted in horizontal strips from bottom up with an advance of four to five square-sets with respect to the subsequent slice.

When the size of the ore body along the strike is considerable and ore production must be increased the slices can also be mined additionally from the flanks towards the centre of the deposit. In this case a pillar is abandoned between the central and flank sections of extraction and is then recovered during the second mining in short blocks from top down.

Development workings in vertical slicing are: a lateral or production drift in the foot wall; a production drift in the hanging wall; crosscuts driven along the boundary separating two slices, and three-compartment raises put up in the ore near the hanging wall, along the slice boundaries.

The raises serve for the passage of men, the dumping of mine-fill, the delivery of timber and equipment from the upper horizon down to the productive slice.

The ore in each horizontal strip of a vertical slice is broken out across its entire width by shooting shallow horizontal holes. After

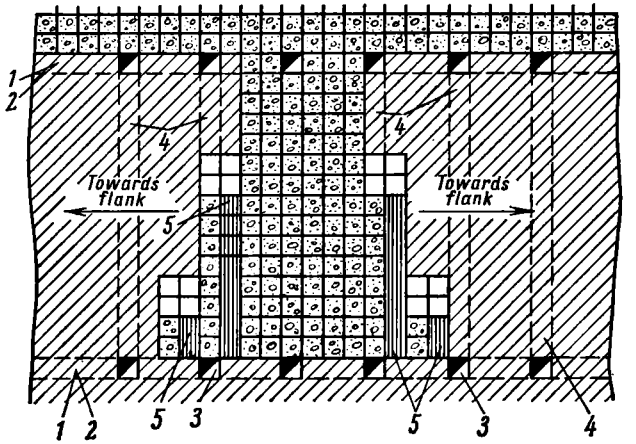


Fig. 204. Mining in vertical slices equal to two square-sets in width (vertical section of a thick ore body on strike):

1—lateral drift in foot wall; *2*—productive drift in hanging wall; *3*—crossdrift; *4*—raise in hanging wall; *5*—ore-chutes arranged after every three square-sets across the strike

firing a round of holes and cleaning of ore the mined-out space should be immediately supported with square-sets. The broken ore slides down the extensible ore-passes and proceeds onto the haulageway via the chutes.

Methods of Mining in Blocks with Downward Stoping of Ore, Artificial Support and Filling

General. These methods find application with extremely unstable, shattered ores which do not permit the undercutting of the solid mass in place even over a small area, and also when the enclosing rocks and the ground surface have to be maintained. Their advantages are low ore losses (the ore does not spill into the mine-fill) and safety of operation. In stoping it is very important to exercise rigid control of the observance of the elevation marks of individual slices, proper timbering and careful preparation of tight flooring, and also of timely and compact packing of the worked-out slice.

The mining methods with support and filling and downward stoping may be classified into methods where filling is done after the whole block has been taken out, and methods where filling is done concurrently with the extraction of ore in a slice. In both cases the stope is timbered right after the extraction of the ore. The former method is rarely used. A brief description of the latter method is given below.

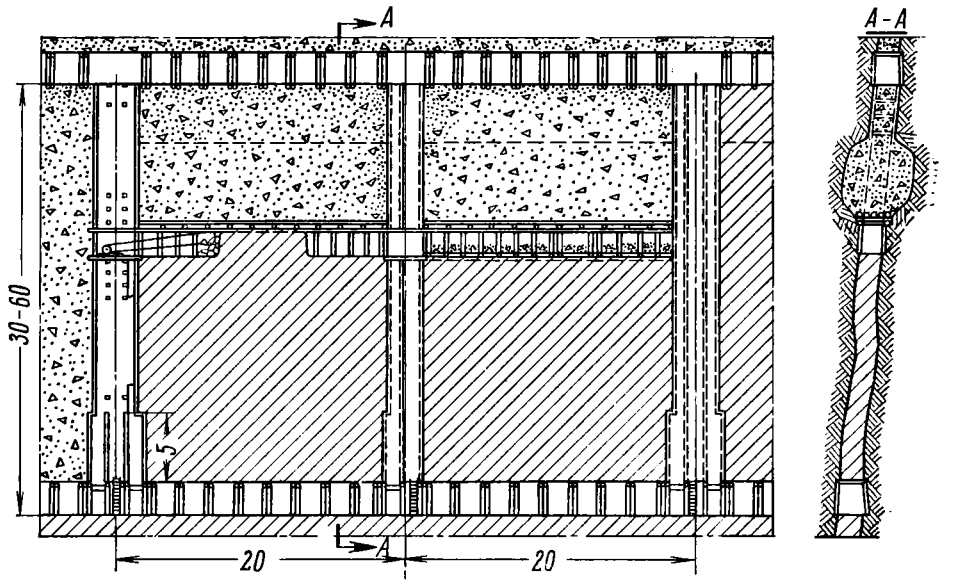


Fig. 205. Downward mining with a continuous face, full-set timbering and slashed rock filling

Downward Mining with a Continuous Face, Full-Set Timbering and Slashed Rock Filling (Fig. 205). This method has been tested in the field. A block is prepared by the excavation of drifts and raises. Stoping is started from a raise by breaking down a slice of ore in the floor of the top haulageway. The broken ore is slushed to the ore-dump compartment of a raise. Upon extraction of the first slice the stoped-out space is filled with pack. This is placed on a timber flooring laid down directly on the ground or on struts. The second slice is taken out underneath the flooring. After the second slice has been extracted a flooring is laid down on its bottom onto which the fill from the first slice is passed. The latter is then refilled with waste. The third and subsequent slices are mined out in the same way as the first two, but with packing of the stope area by ripping

the host rocks of the hanging and foot walls. The host rocks are broken down with the aid of shallow holes drilled upwards at an angle of 35-45°. This system ensures safe working conditions and practically complete extraction of the mineral.

Limited face room is an important drawback of the system.

An analogous variant of the system has been applied at a tin mine in the Maritime Territory of the USSR with enclosing rocks exposed over the height of two slices. In working out a vein from 0.5 to 2 m thick the output per faceman reached 2-2.2 cu m per shift, timber consumption was 0.12 cu m per 1 cu m of ore, explosives consumption 2.5 kg/m^3 and dilution 29%.

Downward Mining in Blocks with Artificial Support and Filling. The application of this new method became possible after the successful introduction of self-cementing fill. Its main features are as follows (Fig. 206*a*). Ore in a level 30 m high is taken out in blocks along the strike measuring 30 m in width. Development workings within the block include two haulage openings on the main floor and two raises in each block. Subsidiary development in each slice involves driftage on the block boundaries of two openings—a scraper drift 10.5 sq m in cross-section and a 7.5 sq m rock-fill raise. Slice crosscuts are run at different elevation marks so as to ensure an angle of cut of 10-15°, i.e. equal to the angle at which self-cementing fill flows freely. The pack moves by gravity and fills the cut up to its back. Stoping is done through cuts made to the rise in the direction from a slice scraper drift towards a slice waste roadway. Within a slice stoping proceeds in the general direction from one flank to the other. The first (top) slice is taken out with the use of reinforced support and lagging of the back. After two cuts have been extracted the first cut is boarded off, lined with a fabric and filled with self-setting pack. Then stoping is started in the third cut; when it is completed the second cut is lagged off and filled. Then the operations are repeated in the same sequence with the maximum number of simultaneously filled cuts limited to three. Driftage of slice roadways in the underlying slice is started after the extraction and packing of the top slice have been completed. Experience shows that with good-quality fill the pack mass acquires sufficient strength (up to 60-70 kg/cm^2) in 2-3 weeks after the termination of filling. The adopted cut section is 10.5 sq m (width 3.5 m, height 3 m). The ore beneath the fill is broken by drilling and blasting of holes 1.5-2.0 m deep. The ore is slushed with small-capacity (up to 0.45 cu m) scrapers. With stable pack the cuts are supported with props blocked directly against the fill; with unstable pack they are wedged against sill pieces or girders laid on the bottom of a superjacent slice before it is filled (Fig. 206*b*). Metal netting is used for lining. With good-quality mine-fill the solid mass of pack in a slice up to 3-3.5 m

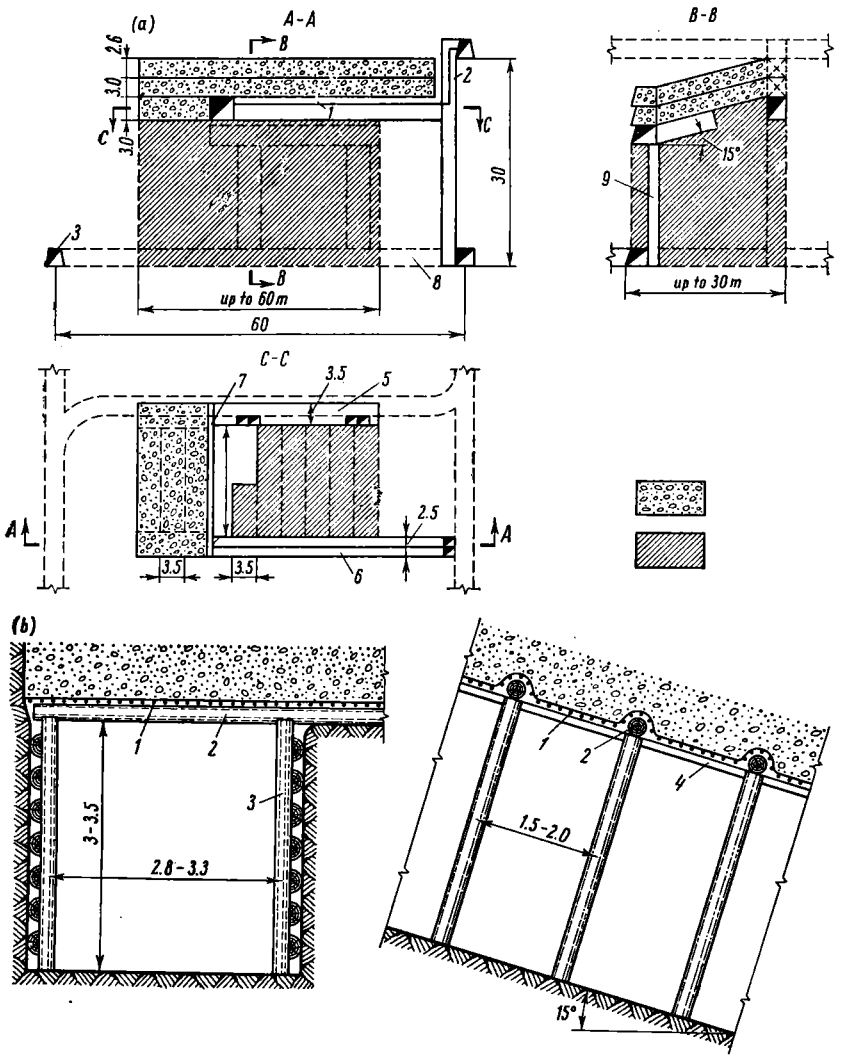


Fig. 206. Mining a block of ore by downward slicing with timbering and self-cementing fill:

a—general layout: *1*—flushing pipeline; *2*—ventilating-and-stowing raise; *3*—haulage cross-drift; *4*—cut; *5*—slice drift; *6*—slice stowing drift; *7*—board lagging in cuts; *8*—haulage drift; *9*—ore-chute; *b*—timbering in cuts: *1*—metal wire-netting; *2*—girders; *3*—prop; *4*—stull

high makes up a monolithic and stable structure. Experience indicates that an artificial roof without support may be exposed over 7-8 sq m, the aggregate area of unpacked but artificially supported space reaching 100-150 sq m. In working under an artificial roof constant control of its stability is of paramount importance.

In addition to visual observations the stability of an artificial roof is checked by measuring rock pressure manifestations with the aid of hydraulic capsules, and roof deformations by using shallow-sunk metal reference pegs and an extensible graduated survey stake. Observations at one of the mines revealed that at the time stoping in a slice was completed the roof had subsided by 1.5-1.8 mm. Permissible exposure areas in an artificial roof may be determined by calculations proceeding from the following basic considerations.

1. The layers of self-setting fill are not strongly bound with each other and may be considered as free or jammed slabs.

2. When computing the supporting power of an artificial roof the weight of the last self-cementing fill layer only should be taken into account.

3. Theory allows substitution of a thick beam for a slab in calculations, bearing in mind that the ratio of the least linear dimension of the exposure to the thickness of the deposit is always less than ten.

4. A beam may cave in either under the effect of tangential stresses in a section lying close to the point of support, or under the effect of compressive stresses. In calculating the beam the effect of both these stresses should be considered.

A calculation made with due consideration for these points, given the volume weight of the pack mass (2.1 tons/cm^3), the height of the slice ($h = 3$ m), the width of the exposure, i.e. the beam length ($l = 2.5$ to 20 m), the safety margin ($n = 2$), and the internal friction angle ($\rho = 25°$) showed that the exposure of the pack mass should not exceed a span of more than 10-12 m, the strength of the solid fill to be at least 15 kg/cm^2. A simple unit as shown in Fig. 207 may be used for the preparation of self-setting fill. The fill is passed down via boreholes lined with case-pipes 120 mm in diameter. There are two such boreholes, one of which is active and the other a stand-by hole.

Among the advantages of downward mining with self-cementing fill are: complete extraction of the mineral (there are no pillars), elimination of losses in the pack, and the possibility of selective stoping. The yield per block with this system reaches 500 cu m per month. Output per faceworker (less filling) is up to 5.5 cu m per shift, timber consumption for support 0.03 cu m per 1 cu m of ore; and explosives consumption 0.5 kg/m^3. The net mining cost in this method is 10-15% less than in upward slicing with filling. The

technological potentialities of the system are far from exhausted (the use of metal support, extraction in twin cuts and longwalls). The system can be applied in mining a disturbed mass of solid ore and abandoned pillars. It acquires special importance in taking out valuable and self-igniting ores and where the ground surface and the solid mass of enclosing rocks have to be kept intact.

Downward horizontal slicing of ore with filling is also successfully employed at one of the Japanese copper mines. An ore body

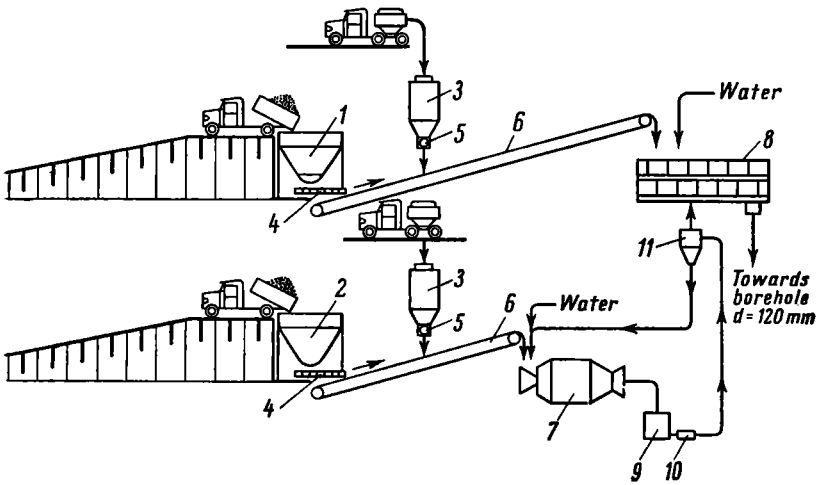


Fig. 207. Flow-sheet of a stowing set with an output of 25 m³/hr of ready-made mix:

1—4 cu m sand bin; *2*—4 cu m slag bin; *3*—10 ton storage bunker for cement; *4*—vibrating feeder; *5*—discharge opening; *6*—conveyer; *7*—ball mill; *8*—mixer for self-cementing mine-fill; *9*—1 cu m surge tank provided with a propeller-type agitator; *10*—slime pump; *11*—hydraulic cyclone

20 m thick is mined in blocks 30 m long each. A block is stoped out in slices 2.5 m high worked through crosscuts 3 × 2.5 m in cross-section which are pushed forward from a slice drift. Chain pillars 3 m thick are left between crosscuts, and once the crosscuts have been packed the abandoned pillars are robbed with subsequent filling of the mined-out space. The building-up of a pack mass begins with laying down of wire netting and steel reinforcement on the bottom of a slice. This is followed by laying a layer of lime 0.5 m thick. After a lapse of several hours the remaining space is filled with sand and crushed slag supplied through a pipeline. The water from the hydraulic patch is drained through a bulkhead lined with cotton fabric. Crosscuts are supported with light timbering (props and stull-pieces). As a result each slice has artificial slabs 0.5 m

thick covered with a hydraulic patch (sand and slag). Mobile loader-haulage units are used for loading and transporting ore in the stopes. From 4 to 5 kg of steel bars and netting is consumed per 1 sq m of the slice area. Metal strips are laid down along the cut; the netting has a mesh size of 50 mm. Output per faceworker reaches 20 tons per shift.

Methods of Mining in Rooms Along or Across the Strike by Horizontal Slicing with Square-Set Stoping Plus Filling

Room mining on strike is practised in ore bodies 4-10 m thick, and across the strike in working ore bodies whose thickness exceeds 10 m. The other conditions are the same as in all systems with timbering and filling.

In the systems under review the sequence of operations in the rooms is analogous.

Below we describe room stoping across the strike.

Room Mining Across the Strike by Horizontal Slicing with Square-Set Stoping and Filling. A deposit is worked in levels with a height interval of 30-40 m. If the interval is larger, the level is divided into sublevels. Each level is broken up into blocks 12-24 m wide, their length being equal to the thickness of the ore body. A block consists of a room and a pillar. With unstable ore and wall rocks and a high level interval the width of the room diminishes while that of the pillar increases correspondingly. The room width often equals that of the pillar. Rooms are worked out first. Pillars are recovered by the same method, but this is done from top down when the ascending order of extraction is hazardous. Intervening pillars can be recovered by other methods: in short blocks or by vertical slicing with timbering and filling.

The development workings in the horizon (Fig. 208) are either a lateral or a reef drift in the foot wall of the deposit. In working very thick ore bodies the reef drift is excavated in the hanging wall. Crosscuts are commonly run along the room axis, and raises in the hanging wall. For pillaring purposes crossdrifts and raises are driven additionally. Ore-chutes made in rooms on the boundary with intervening pillars sometimes serve as raises. Drifts and crosscuts are supported by square-sets, their width being equal to one or two square-sets.

Ore in rooms is taken out by downward horizontal slicing, starting with the first or second slice. Stoping usually proceeds in the general direction from the hanging to the foot wall. The ore is often extracted by the overhand method, which enables it to be moved downwards with pack disposal by gravity. To preserve the haula-

geway openings the zero and often also the first slice is temporarily left untouched. They are worked out during the extraction of the subjacent horizon, or simultaneously with intervening pillars.

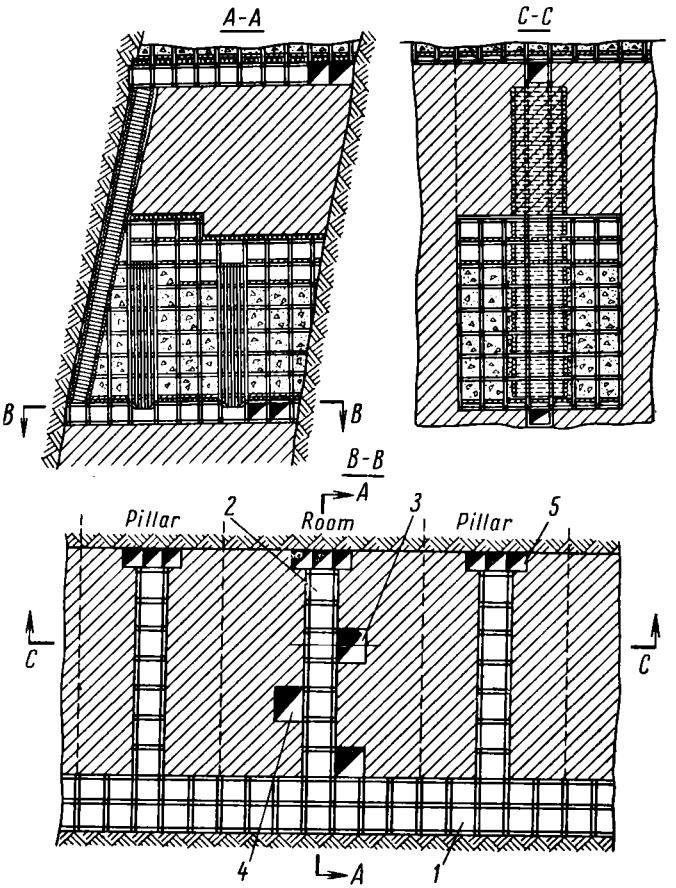


Fig. 208. Square-set stoping and filling by rooms across the strike with horizontal slicing:
1—foot-wall drift; *2*—crossdrift; *3*—ore-chute; *4*—ladderway; *5*—three-compartment raise

Stoping operations in a slice are started from a raise or from a special uphill working equal to one slice in height. To break the ore in a stope, horizontal holes are drilled with jackhammers or drifters. In a sufficiently stable ore up-holes are drilled with stopers. The holes should be deep enough to ensure a stope face advance

over a distance equal to one square-set. Prior to breaking of ore a flooring is laid down upon the fill of the bottom slice, which is withdrawn after the breakage and cleaning of ore in the slice. The flooring on the bottom of the first mining slice is made of slabs 100-150 mm thick or of round lagging covered with 50 mm boards. This flooring is not removed, but covered with pack. After the ore in the stope has been broken down timbering must be set up immediately.

The broken ore is delivered to draw-holes, which are extended every 6-8 m, either by hand (using shovels or small mine-cars) or by small scraper hoists.

Packing material from a raise is brought to the mined-out slice in the same manner. Before filling a slice the walls of the room should be lagged.

Because of its rare application square-set breast-stoping and filling is not discussed here.

Advantages and Shortcomings of Mining with Artificially Supported and Filled Stope Area

The basic merits of such systems are: the possibility of protecting the ground surface and enclosing rocks against caving, provided the pack is tight enough; insignificant losses and dilution of ore if a solid flooring and protective side walls are available in the room; the possibility of using the systems in extremely unfavourable conditions of mining (intricately shaped ore bodies, frequent changes in the mode of occurrence, very unstable ore and host rocks); the possibility of underground sorting and delivery of different grades and classes of ore; operations can be carried on concurrently in several levels, this ensuring high mining intensity.

Among the basic disadvantages of the methods are: low output per faceman; considerable timber consumption and, consequently, high cost of stoping, the net cost of ore being higher than with other systems; fire hazards owing to the large consumption of mine-timber.

The effectiveness of mining with timbered and filled stope areas can be increased appreciably by all-round mechanization of drilling, haulage of ore and fill, proper timber framing, application of multiperforator and multiface methods, utilization of new types of support, gunniting of the upper layer of the mine-fill, instead of laying down floorings, and introduction of downward successive slicing with self-setting pack. The use of self-setting pack and mobile equipment is one of the most important factors in raising the efficiency of this method.

Technical and Economic Characteristics of Mining with Artificially Supported and Filled Stope Area

The basic characteristics of such systems as applied to frequently encountered conditions of mining ore bodies are: output per faceworker 4-6 tons per shift; coefficient of ore recovery 0.93-0.95; dilution factor 0.1-0.03 (depending upon the thickness and shape of the ore body); timber consumption 0.04-0.06 cu m/ton of ore (including timber consumed for flooring and lagging); explosives consumption 0.2-0.3 kg/ton of ore.

The labour efficiency figures may be increased considerably by taking the measures listed in the foregoing text; for instance, in descending slicing with self-cementing fill and the use of mobile loader-haulers the output per faceworker may be raised to 20 tons per shift.

5-7. METHODS OF MINING WITH CAVING OF HOST ROCKS (CLASS VI)

General

These systems are widely used in the mining of ore bodies. They are employed as independent methods in working individual ore bodies and as auxiliary ones for the recovery of pillars and ceilings abandoned when using other systems of mining.

The proportional share of the basic system in this class—topslicing—in the mining of Soviet ore deposits amounts to 25% in the case of copper ores, about 20% for polymetallic ores, about 85% for alumina, and 2 to 5% for gold ores.

These methods find extensive application (80-100%) in working gently dipping thin iron- and manganese-ore deposits.

The restricted application of the methods in the gold industry is largely due to the inconsistency of the actual mining conditions with the parameters of the methods. They are also rarely used in working thick iron-ore deposits, mainly because of the relatively high mining cost of ore in topslicing as compared with other mining systems, such as sublevel and block caving.

The conditions for the use of mining methods with caving of host rocks are: unstable and readily caving enclosing country rocks; the ground surface and enclosing rocks may be allowed to settle or cave in the absence of surface structures and nearby water bodies or rocks with a high water content (in the presence of water bodies or high-water-content rocks caving is possible after the diversion of the stream or appropriate drainage); the pitch angles may vary, but steep angles are preferred; ore bodies 3-4 m and more in thickness, including very thick deposits; ore of high value, since methods invol-

ving the caving of host rocks require the consumption of large quantities of mine-timber and the efficiency of drilling and blasting operations is limited; the absence of considerable inclusions of gangue and different grades of ore, since underground sorting and grading of ore in systems with caving of host rocks presents some difficulties.

Methods of mining with caving of host rocks can be applied in working ore bodies of complex configuration with ores of different stability and strength, but ore bodies of regular shape consisting of soft and medium-hard ores lend themselves more readily to this mode of winning. With an ore body of very complex shape it becomes necessary to artificially rectify the contours of the ore body to be mined by extracting additionally strips of barren rock (to ensure normal settling of the timber cushion). In this case one must make an economic comparison of topslicing and methods with artificial support and filling, which can also be done in the working of a complex-shaped ore body without any need for its rectification.

Class VI includes the following methods of mining:

by blocks with slicing of ore (taking out several slices)—by stoping through small-section cuts (extraction by splitting), by longwall stoping;

by pillars (long blocks) with single-slice stoping of ore—by extraction through cuts (splitting), by longwall mining.

Methods with caving of host rocks may be practised both with productive development and with the use of stone workings.

Mining in Blocks with Slicing of Ore and Caving of Host Rocks

Variant of Ore Stoping through Small-Section Cuts. Topslicing is applied under the mining conditions mentioned above. The system is commonly used in working thick deposits, from 6-8 m and thicker. Small thickness (from 3-4 to 6-8 m) is no obstacle to using the method, but it increases development work, the losses and dilution of ore. Under favourable mining conditions and with proper conduct of operations losses and dilution do not exceed 3-5%, this being one of the principal merits of the system, especially with ores of high value. Topslicing can be successfully employed in working areas of caved or fissured ore and in such conditions it is indispensable. A detailed description of topslicing and its versions is given below.

With topslicing, an ore body in a level is taken out by individual blocks from 20 × 20 to 50 × 50 m in size. In deposits of limited thickness blocks up to 40-50 m long are extracted across the entire thickness of the ore body. Blocks are mined from the hanging to the foot wall, more often from the flanks to the centre of the ore

body (Fig. 209). The sequence of extraction is greatly influenced by the conditions attending topslicing and cover caving, e.g. it depends on which part of the deposit lends itself most readily to the caving of cover rocks, account being taken of their thickness and stability. Complete and uniform caving of host rocks guarantees successful application of the topslicing method. In the absence of favourable conditions measures should be taken to induce the caving

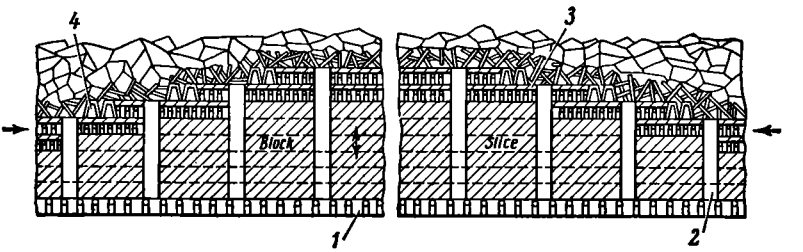


Fig. 209. Mining of blocks in a level with stoping from the flanks to the centre of the ore body:
1—haulage drift; *2*—block raise; *3*—slice drift; *4*—cut

of cover rocks by drilling and firing shallow or blast-holes in the back of the mined-out space, or by blasting explosive charges in chamber workings driven in the enclosing country rock. Uniform caving of host rocks and reduction in the contamination of the ore with waste are achieved by employing a timber cushion. Because of the difficulty of ventilation and supply of materials to the working faces the level interval in topslicing should not exceed 40 m. It may be increased when stone workings—roadways and raises—are excavated in the foot wall, thus ensuring communication with the top horizon. The ore in a block is extracted by downward slicing. For safety of operation in a block each overlying slice must be mined with a lead of not less than 15 m over the subjacent one.

In topslicing, the development openings are a productive drift or crosscut, and a raise. In an ore body up to 15-20 m thick the productive drift is pushed forward along the foot wall contact (unilateral mining in the block) (Fig. 210*a*). With an ore body exceeding 20 m in thickness the productive drift is usually run in the middle of it (bilateral mining in the block). In working very thick deposits parallel productive roadways are driven every 20-40 m, or either one productive or one lateral drift is excavated in the foot wall with crosscuts driven up to the hanging wall (Fig. 210*b*). In working ores presenting fire hazards (cuprous pyrite and chalcopyrite) it is advisable to resort to development with the drivage of a

fringe lateral drift, stone crosscuts and raises. Development of very thick chalcopyrite deposits is done by running two lateral

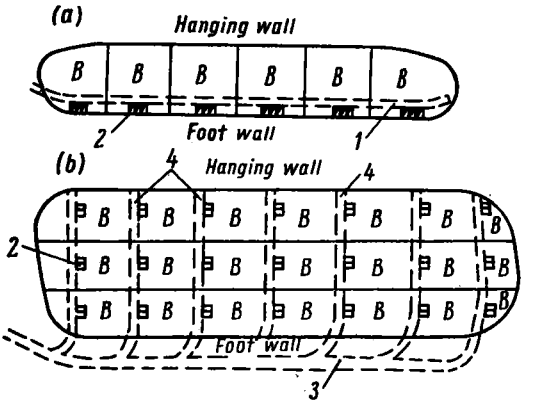


Fig. 210. Disposition of blocks in topslicing (plan view):
a—with an ore body of limited thickness (development through drifts); *b*—with an ore body of considerable thickness (development through crosscuts with a lateral drift): *1*—productive drift; *2*—block raise; *3*—lateral drift; *4*—crosscut; *B*—blocks

drifts with stone raises (in the hanging and foot walls) to permit preventive silting of stoped-out blocks on both sides.

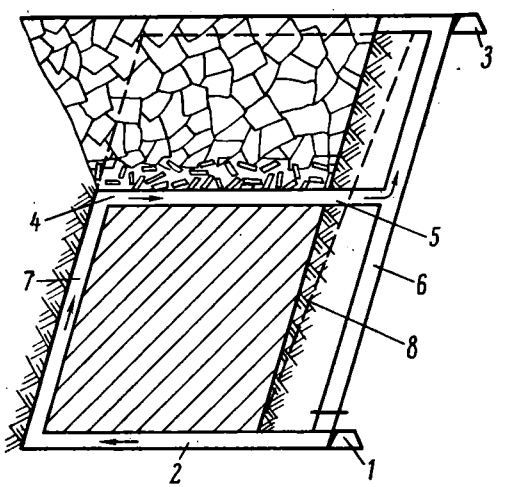


Fig. 211. Topslicing in a block by using stone workings (vertical section across the strike):
1—stone haulage drift; *2*—cross-drift; *3*—lateral vent drift; *4*—production slice; *5*—breakthrough; *6*—stone raise; *7*—raise in ore; *8*—boundary of zone of rock displacement along the foot wall; → direction of air current

Development with the use of stone workings (Fig. 211) is more costly, but it allows drift maintenance expenses to be cut down, particularly with weak, soft ore; it also helps set up an efficient venti-

lation system without employing individual fans in each block, facilitates the delivery of supplies, communication with individual slices, and control of underground fires.

The raises have three compartments: a ladderway for the passage of men, a service compartment for lifting mine-timber and tools, and an ore compartment for dumping the mineral. Sometimes an air compartment is also made, but more often air currents are passed through the ladderway and service compartments. Raises in the level are spaced in accordance with the accepted size of blocks and the methods of ore haulage therein. A smaller block size makes for a greater intensity of mining of an ore body due to the larger number of active blocks. A greater block size enables the scope of development work to be reduced, but the rational distances of scraper hauls of ore in a slice restrict the size of the block down to 20-25 m with unilateral and 40-50 m with bilateral haulage of ore.

Upon completion of development work drivage of temporary workings is started; these include slice drifts or productive crosscuts. When driving the latter a thin ceiling is sometimes left in the back so as to avoid direct contact with the mat. The abandoned thin ceiling is recovered later during the subsequent extraction of cuts.

A slice drift or crosscut is run to provide ore reserves ready for extraction.

This is followed by stoping, which is done by individual cuts with a small-section fore-breast in the direction from the block boundaries towards the raise. Cuts 2.5-3.0 m wide and 2.5-3.5 m high are driven from the slice drift of the slice crosscut to the block boundaries, with the ore broken by firing shallow holes, portions of the cuts being timbered with frame-sets. The length of a cut varies within a wide range—from 3-4 to 20-25 m—and depends upon the size of the block and the methods of ore haulage. The ore broken in a cut is slushed straight (Fig. 212) or at an angle to the ore compartment of the raise (Fig. 213).

The first method of haulage is employed rather rarely, since the raises should be closely spaced; the second method is more complicated, for it requires slushing in two directions or arrangements for turning the scraper. For this reason a third method of ore haulage is commonly used, which involves the utilization of storage workings (Fig. 214).

Excavation of storage drifts or crosscuts does not involve any additional expenses, since later on they serve as slice workings. Storage openings greatly facilitate scraper haulage and improve conditions of ore transportation. Scrapers, both in the mined slice and the storage working, operate along a straight path; besides, the necessary stock of ore is built up in storage workings which is suf-

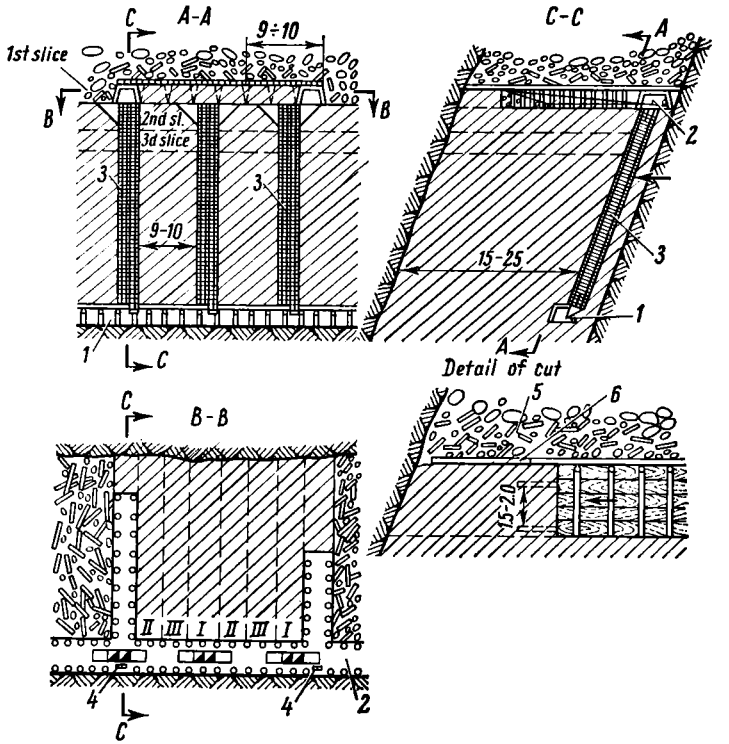


Fig. 212. Topslicing with direct slushing:

1—haulage drift; *2*—slice drift; *3*—raise; *4*—scraper hoist; *5*—mat; *6*—flooring; *I*, *II*, *III*—cuts excavated from the foot to the hanging wall

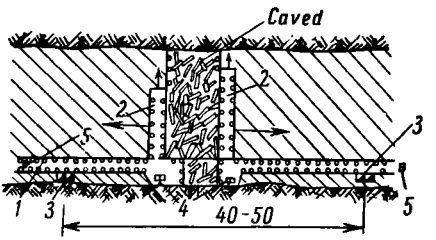


Fig. 213. Topslicing with ore slushed at an angle:

1—slice drift; *2*—cut; *3*—raise; *4*—hoist for slushing ore in the cut; *5*—hoist for slushing ore in the slice drift; long arrow indicates general direction of stoping in block

ficient to load a train of mine-cars. This is especially important in extracting the lower slices of a block. In stoping the upper slices a raise may serve as a storage working.

After the ore has been completely extracted in a cut its floor is covered with a flooring made up of round lagging, substandard boards, etc. The flooring is laid down lengthwise so that it can be easily caught up by frame-sets when a lower slice is being mined.

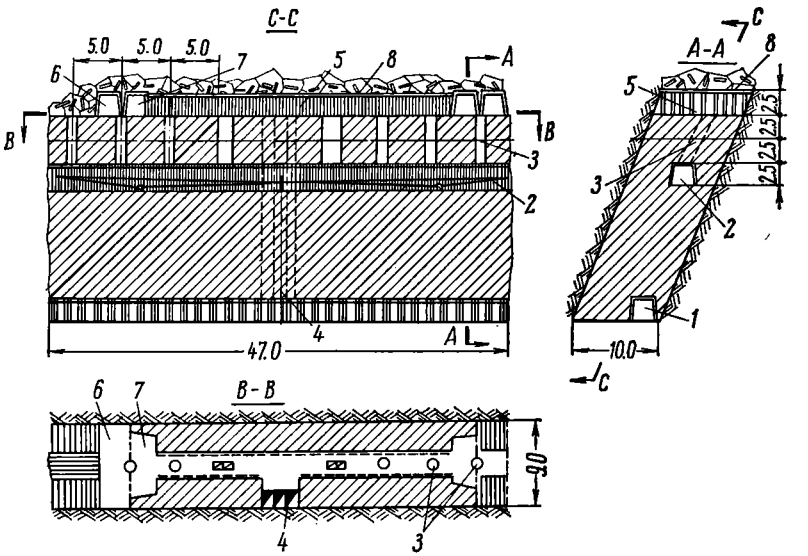


Fig. 214. Topslicing with storage drifts:
1—haulageway; *2*—storage drift; *3*—ore transfer pass; *4*—raise; *5*—slice drift; *6*—worked-out cut; *7*—cut in the process of mining; *8*—mat

In the cuts of the first three to five slices timber is added to the flooring to form the necessary cushion at least 2-3 m thick. After the flooring has been laid down the cut is caved; this is done by piercing timber-set props with hand or power augers, charging the holes with small amounts of explosives and blasting the entire timbering by means of a detonating or electric fuse. Timely and complete caving of a cut is of prime importance for safe conduct of operations and reduction in ore dilution. The latter is considerably favoured by the presence of proper flooring and cushion, and by the lagging of the side walls in the stoped-out cut (on the boundary line with the cut to be mined) with slabs or boards.

Owing to the concentration of large amounts of timber (flooring, mat) the topslice method of mining presents great fire hazards. To

reduce this, wire-netting cushions and metal posts are widely used in Soviet coal mines.

Topslicing with the use of metal posts and wire-netting is regularly tested at metalliferous mines of the USSR. The efficiency of using such posts and mat is proved by the following figures recorded during pilot operations: a 25-30% rise in the output per faceworker, a 50% reduction in timber consumption, a 40-45% drop in stoping cost as compared to topslicing with wooden timbering and mat.

Apart from lessening fire hazards the use of metal posts enhances the safety of operations, since metal posts can bear much greater loads than timber props can.

The following additional measures should be taken to ensure fire safety in topslicing: the installation of special fire-fighting water piping on the main horizon and in the slices being extracted, the use of enclosed light sources (storage battery lamps), keeping electric wiring system in proper condition, good ventilation of stopes, prohibition of smoking in the mine and strict control of the implementation of fire-prevention measures such as the observance of the stipulated temperature limits.

In mining chalcopyrite deposits it is recommended that lateral drifts and raises in the foot wall be excavated, which greatly facilitates fire control. Lateral workings help ensure good ventilation and arrange sand-and-clay pulp supply to worked-out cuts through special pipelines in order to lower the temperature therein. In the absence of lateral workings artificial ventilation should be set up in each block; to achieve this, air is piped into cuts being extracted by means of special fans installed on the haulage level.

A variant of topslicing with the extraction of ore through single cuts has been considered above. In recent years a number of Soviet mines have been using to advantage a version of this system involving simultaneous stoping of ore in a slice by twinned cuts, which raises considerably the tonnage yielded by a block and improves the efficiency of stoping operations. The basic features of mining a slice by twinned cuts are evident from Fig. 215. The topslice method is much more effective than the methods with artificial support and filling, and for this reason the latter are often replaced by topslicing. The technical and economic characteristics of this system are listed in Table 24, p. 426.

The introduction of integrated production teams may greatly increase labour efficiency in topslicing.

Variant of Topslicing with Longwall Stoping of Ore. This variant (Fig. 216) is intended to increase the effectiveness of the topslicing method in the mining of ore deposits. The essential point of distinction between this variant and the usual one discussed above is that stoping is done by carrying a face of considerable length.

Longwall mining enables drilling and other operations to be carried out over a large stope face area. It also aids in implementing a number of measures aimed at raising the efficiency of the topslicing methods, such as: (1) excavation of square-setted working trenches (Fig. 217) replacing raises and intended for lowering and lifting labour force to and from an active slice, for hoisting mine-timber

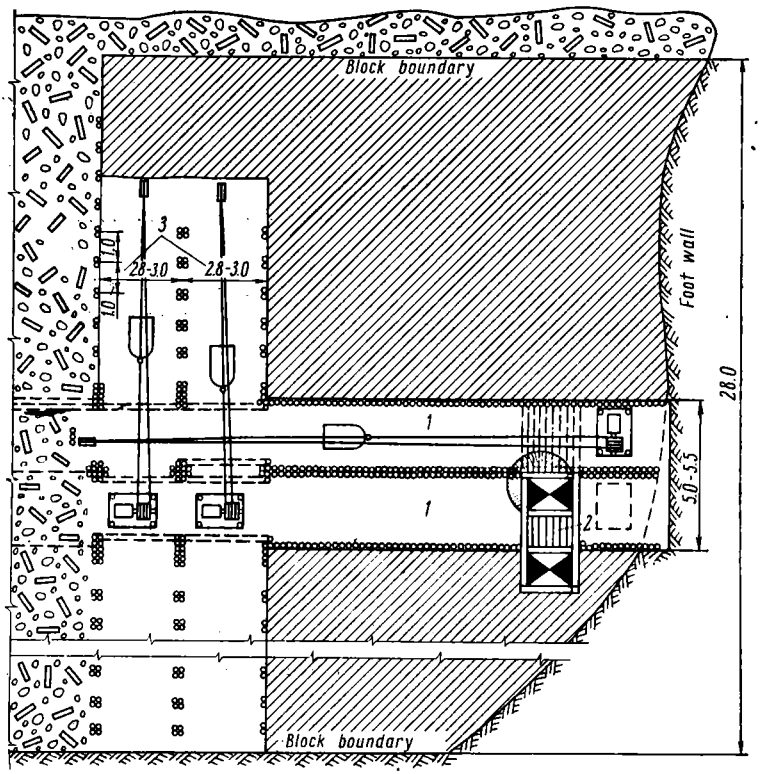


Fig. 215. Slice extraction by twin cuts (plan view):
1—slice crossdrift; *2*—raise; *3*—twin cut (in the stage of mining)

and dumping broken ore via lagged-off compartments, and for the passage of air; (2) the use of metal support; (3) the use of more powerful scraper hoists and larger scrapers.

Mining experience at a number of chalcopyrite mines in the Urals and at Soviet bauxite and polymetal mines has demonstrated this variant to be highly effective. The output per faceworker has been increased by 40-60%, the yield per block by more than 100%, losses

of ore have been cut down and the prime cost of stoped mineral greatly reduced. Practice has proved that the basic prerequisite for the successful application of this system is high mining intensity, for otherwise longwalls may suffer sudden collapses.

Variant of Topslicing with Flexible Metal Flooring (or Roofing). This is a new version of the method. It consists in the following. Ore

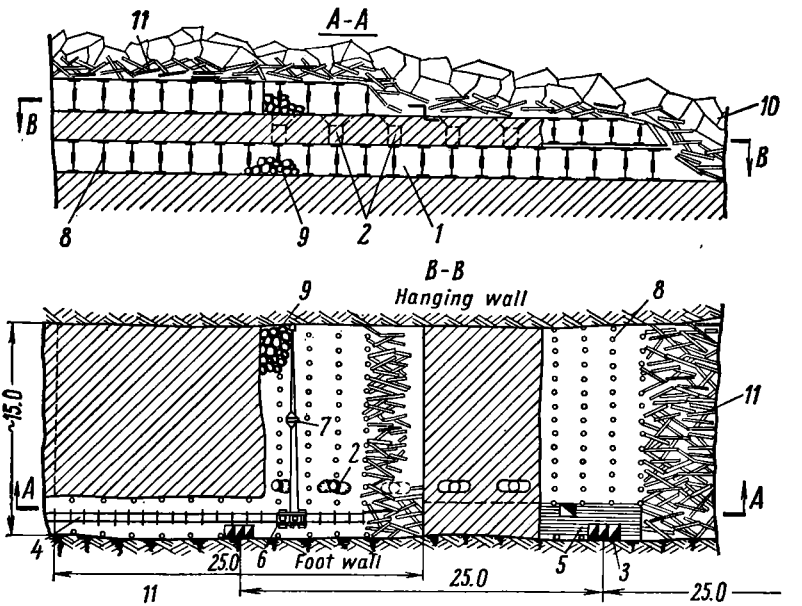


Fig. 216. Longwall stoping of ore in topslicing:
1—storage drift; *2*—pull holes; *3*—raise; *4*—rail track; *5*—flooring; *6*—scraper hoist; *7*—scraper; *8*—metal posts; *9*—broken ore; *10*—mined-out space with caved roof; *11*—mat

in a slice is extracted under the protection of a metal flooring mounted in the upper assembly slice. It is made up of iron strips laid down in parallel crosswise rows and spaced at 0.3 m. The iron strips are covered with a 25 × 30 mesh wire-netting (Fig. 218). The netting, up to 1.5 m wide, is decked in two or three rows. The ore in the assembly slice is taken out through cuts supported with timber framesets spaced at 300 m. This leaves a clearance of 50 mm between individual props which accommodates iron strips forming a 200 mm cell. The netting is unrolled along the cuts, slightly overlapping the timber posts. Upon completion of the flooring assembly, cuts stoped out in the block are caved by blasting timber props. Output per faceman in the assembly slice, including installation of flooring, varies from 2.5 to 3.5 cu m per shift.

In mining a slightly sloping deposit the slice adjoining the hanging wall is worked by a long face supported by strain posts with subsequent laying down of metal flooring (an inclined assembly slice in addition to the horizontal assembly slice in the top portion

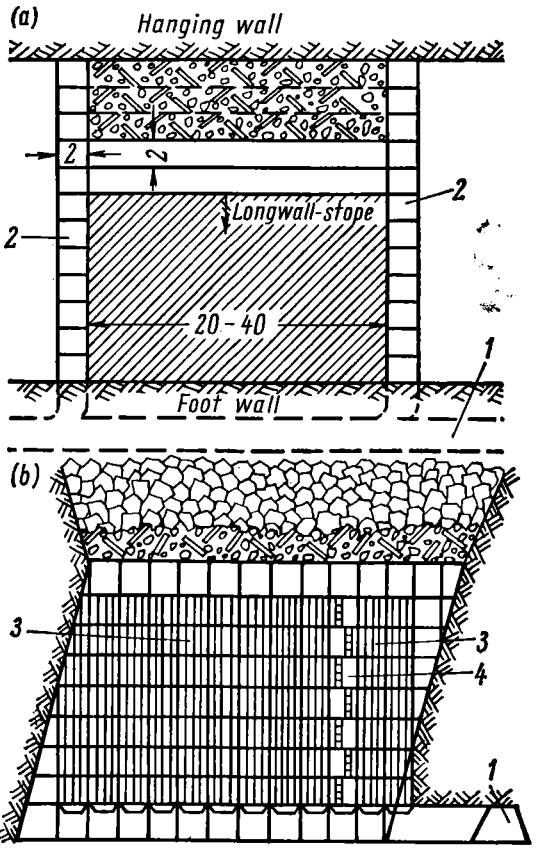


Fig. 217. Topslicing of a block by a longwall with working trench:
a—plan view of slice; *b*—section along trench; *1*—lateral drift, *2*—square-setted trench; *3*—ore-dump compartment of trench; *4*—ladderway compartment of trench

of the block). To ensure safe working conditions under the flooring the hanging wall rock is longholed and blasted when productive cuts in the assembly slice are caved. Slicing under the flooring is done after trenches on strike have been excavated and intertrench stumps abandoned (Fig. 219). The trenches are untimbered, their width in the upper portion is up to 3.5-4.5 m and in the lower portion

1.5 m. They are driven with the aid of holes drilled to a depth of 1.7 m. Intertrench stumps are up to 2.0 m wide at the bottom and

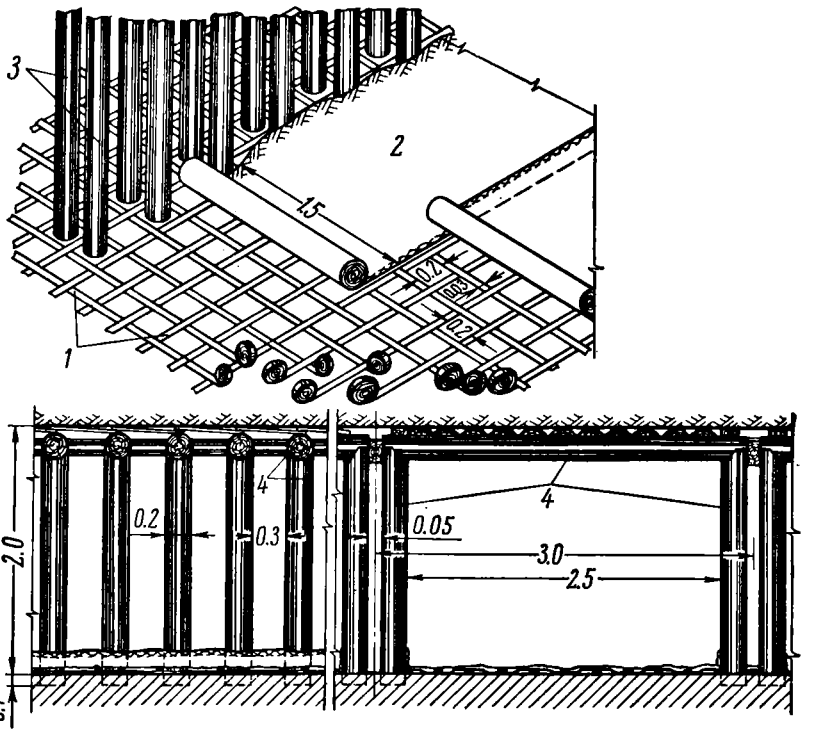


Fig. 218. Support of the assembly slice cuts and flexible flooring laid in the assembly slice:
1—strip iron; *2*—wire-netting; *3*—frame-set post; *4*—frame-set

up to 1.5-1.3 m at the top. They are blasted out along their entire length by means of a detonating fuse. Caving of the flooring onto

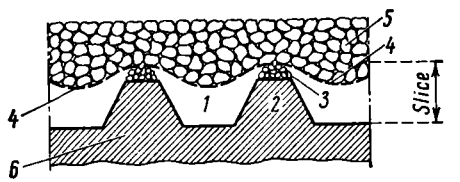


Fig. 219. Trenches and intertrench stumps:
1—trench (along strike); *2*—intertrench stump; *3*—ore caved over stump; *4*—flexible metal flooring; *5*—caved rock; *6*—solid ore in place

the blasted ore of the support intertrench stumps improves the performance of the flooring due to the higher yielding properties

of its base (the rock pressure exerted on the flooring is reduced). In order to raise the strength of the metal flooring it is expedient to run the iron strips diagonally to the trenches. With this variant output per faceman may be set at 2.5-3.0 cu m per shift in the assembly slice and up to 6.5-7 cu m in the underlying slices. A maximum labour efficiency can be achieved in the working of deposits exceeding 8-10 m in thickness. In the variant with flexible metal flooring ore losses may be practically ruled out. The arrangement of flexible metal flooring is illustrated in Fig. 218.

Topslicing with Shield Flooring

In working steep ore bodies (with dip angles exceeding 60°) of regular thickness (from 3 to 6 m) and with unstable host rocks an effective version of topslicing with shield flooring may be employed.

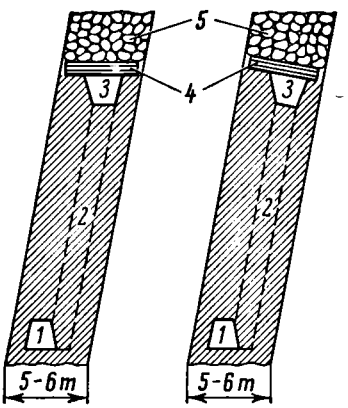


Fig. 220. Transverse section of a topsliced block with shield flooring:
1—haulageway; *2*—raise; *3*—trench (under the shield); *4*—shield; *5*—caved rock

Below we describe this version as applied at copper mines of the Urals. The shield is not sectionalized and consists of three rows of logs laid down across the strike and held together by ropes and bolts (Fig. 220). The shield is assembled in a top slice, where a slice drift is first excavated along the strike. The slice drift is pushed forward from raises located on the flanks of an active block. Drift cuts are run from the slice and shield logs are laid down in them. The stoped cuts are caved by blasting timber props. The assembled shield is brought to a vertical position by blasting out support stumps near a trench driven under the assembled shield. The trench is driven by shooting shallow or long holes drilled from end raises. The stoping involves destruction of support pillars to a height of 1.2-1.5 m, setting the shield down on the fragmented ore and periodi-

cal driftage of the trench under the shield. The broken ore is slushed to a raise (Fig. 220). The technical and economic characteristics of shield topslicing are higher than in other variants of the system. The following figures can be adopted, which are based on the experience at copper mines in the Urals:

1. Rate of progress in depth of mining operations in a block, metres per month . . . . up to 6-8
2. Tonnage yielded by a block, tons per day up to 250-260
3. Output per faceman, cu m per shift . . . up to 7-9
4. Metal consumption, kg per cu m . . . . from 1-2 to 3-4
5. Explosives consumption, kg per cu m . . . 0.7-0.9
6. Ore losses, %% . . . . . from 3 to 6
7. Dilution, %% . . . . . from 3-5 to 6-8

The net cost of stoping is 30-40% cheaper than in an ordinary version of topslicing.

Effective parameters of the shield method are: shield length 20. 30 m, width 3-5 m (1 m less than the thickness of the ore body)- The expenses incurred in making and assembling the shield are covered after it descends to a depth of 6-8 m.

Variant of Topslicing with the Use of Mobile Equipment

In this version small loader-haulage units of various designs are used, such as ПДВ-2 type machines manufactured in the USSR.

When operating the ПДВ-2 unit it is advisable to dispose cuts at an angle facilitating access of this machine into a cut from a slice working. By using mobile equipment and introducing other measures aimed at improving the method (development in stone, drivage of working trenches dividing a block into two half-blocks, stoping of ore in a slice by carrying a long face, utilization of wire-netting as a mat, and installation of metal supports) one may achieve, in working thick ore bodies up to 45 m long, high technical and economic results, for instance:

1. Proportional share of stoping in a block up to 90%.
2. Ore losses up to 3%.
3. Dilution of ore bulk up to 3%.
4. Output per driller up to 70 tons per shift.
5. Output per ore mucker up to 280 tons per shift.
6. Output per faceman up to 23-25 tons per shift.
7. Mine-timber consumption 0.004 cu m per ton.
8. Explosives consumption 0.25-0.30 kg per ton.

9. Metal post consumption 0.0003 piece per ton.
10. Wire-netting consumption 0.095 sq m per ton.

Comparative net cost of stoping is 50-60% of that in ordinary variants with extraction by splitting through cuts, wooden timbering and scraper haulage.

Variant of Topslicing in Gently Dipping Ore Bodies. Topslicing in gently dipping deposits is done by dumping the ore down the raises with scrapers. Topslicing in an ore body dipping at an angle of 18-30° is successfully carried on at a bauxite mine in the Northern

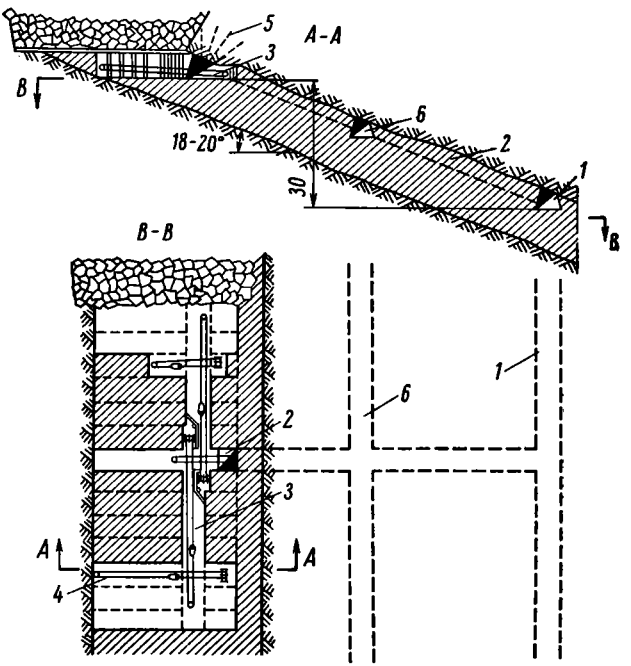


Fig. 221. Topslicing in gently sloping ore bodies

Urals (Fig. 221). A block is developed by running a drift (*1*) along the hanging wall (in conditions prevailing at the above-mentioned mine the foot wall has irregular occurrence and complex configuration). Raises (*2*) are driven up every 40-45 m from the drift. A slice drift (*3*) is pushed forward from a raise, following which cuts (*4*) are excavated on both sides of the block. In a cut and along the slice drift the ore is hauled with scraper hoists of different capacity—7, 14, and 28 kW respectively. The hanging wall is caved after three cuts have been mined out, this being done by firing sectional-

steel holes (*5*) drilled in the roof of the slice drift, and by blasting out posts in the stoped-out cuts. A subdrift (*6*) is excavated in the block for ancillary purposes. By using this method the following results were achieved in the Urals: output per faceman 2.9-3.2 cu m per shift, timber consumption 0.09-0.11 cu m per 1 cu m of ore, explosives consumption 0.9-1.1 kg per cu m of ore, volume of development openings in a block 8-10% of the total ore volume therein.

In addition to the method described the variant of topslicing with stepped long-face extraction of ore on strike is employed at the same mine in working ore bodies up to 4 m thick. A horizon with an inclined height interval of 60-70 m is divided into 2 or 3 benches, the top bench being stoped 3-4 metres ahead of the lower one. The ore in a long face is slushed along the subdrift and an inclined raise. Advance per caving is 9-10 m. The back of a working face is temporarily supported by means of rock bolts with a spacing of 1.1 × 1.1 or 1.2 × 1.2 m. In this variant output per faceman increases to 4 cu m per shift, while expenses on timbering are cut down almost by half.

Pillar Method of Mining (in long blocks with single-slice extraction of ore and caving of cover rocks)

This system is employed in working flat and gently sloping ore bodies up to 3-3.5 m thick, occurring in unstable enclosing rocks. Pillar methods of mining have found wide application in working

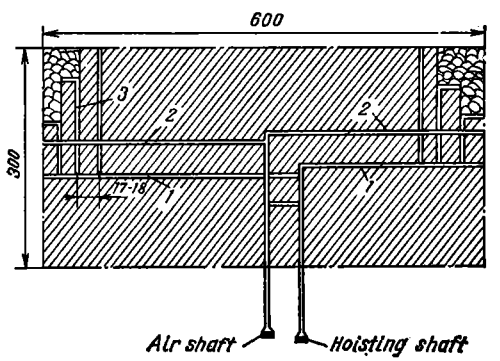


Fig. 222. Opening-up and development of a mine-field in the pillar method of mining

manganese deposits. Two variants of the pillar method are in use: a long-pillar variant with extraction of ore by cuts and a long-pillar variant with stoping by carrying a long face.

Extraction by splitting is practised when the roof is very unstable.

Prior to stoping the mine-field is specially prepared (Fig. 222), i.e. divided by a haulage (*1*) and vent (*2*) drifts into panels 120-150 m

wide. Extraction drifts (*3*) are advanced from panel roadways (every 17-18 m) to the mine-field boundaries. Extraction drifts divide a panel into pillars. The latter are taken out either through single or twinned cuts in the direction from the mine-field borders, stoping in one pillar being done in advance of the other. Figure 223 illustrates the mining of a long pillar by twinned cuts with a temporary ore stump 1.5-2.0 m thick left on the border with the gob area.

Twin cuts are excavated piecemeal with limited application of drilling and blasting, and are supported with frame-sets. In some manganese mines soft ore is taken out by using chipping air hammers.

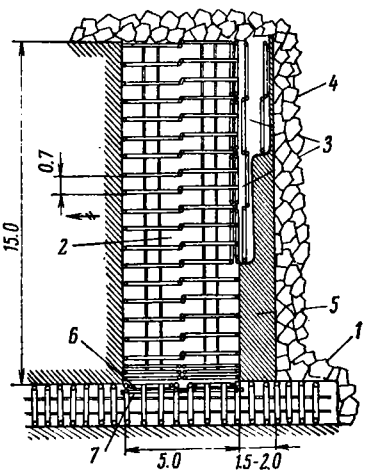


Fig. 223. Long-pillar recovery by twin cuts or blind drifts:
1—extraction drift; *2*—twin blind drift; *3*—short cuts; *4*—mined-out space with caved roof; *5*—stump; *6*—underframe; *7*—room frame-set; arrow indicates general direction of pillaring

When the extraction of the cuts and the stump (the latter is recovered by short cuts) is accomplished the enclosing rocks are caved.

In recent years extraction of ore in cuts has been done by using МБЛ mining combines (Fig. 224), the stoped-out ore being transferred onto a conveyer by means of a reloader. Extraction by narrow splitting (stub headings) is labour-consuming and ineffective, therefore new mine projects envisage stoping with longwalls involving the use of mechanized multipurpose units. When such multipurpose machines are employed mine-fields, panels and pillars are made much larger (mine-fields up to 1.5 km and more in length, panels up to 400-500 m wide, mined pillars up to 400-500 m long and 25-30 m wide). Primary mining and drivage of subordinate development workings in a mine-field is done by excavating panel drifts from main roadways and running subsidiary headings from the panel drifts, with temporary protective pillars up to 30 m wide left near the main and panel drifts (Fig. 225). Drifts should have a yielding metal support made up of sectional members, or a prefabricated reinforced-concrete support. Belt conveyers are used to transport the ore along the drifts and deliver it directly to the ground surface via an inclined shaft. With an annual mine production capacity of 500,000 tons of ore panel drifts should have a finished section of 5.0-5.3 sq m, rough section of 7.4-8.5 sq m, the section of subsidiary

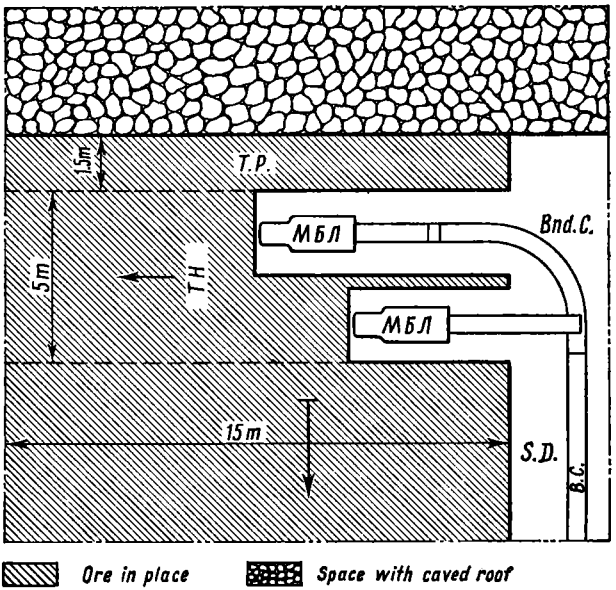


Fig. 224. Pillar recovery by twin blind drifts (pocket-and-fender pillaring): ⟼general direction of pillar robbing;→ direction of extraction of twin blind drifts; *T.P.*—temporary ore pillar; *B.C.*—belt conveyer; *Bnd . C.* —bending conveyer; *S.D.*—shrink drift; *МБЛ*—mining combine of МБЛ type; *T.H.*—twin headings

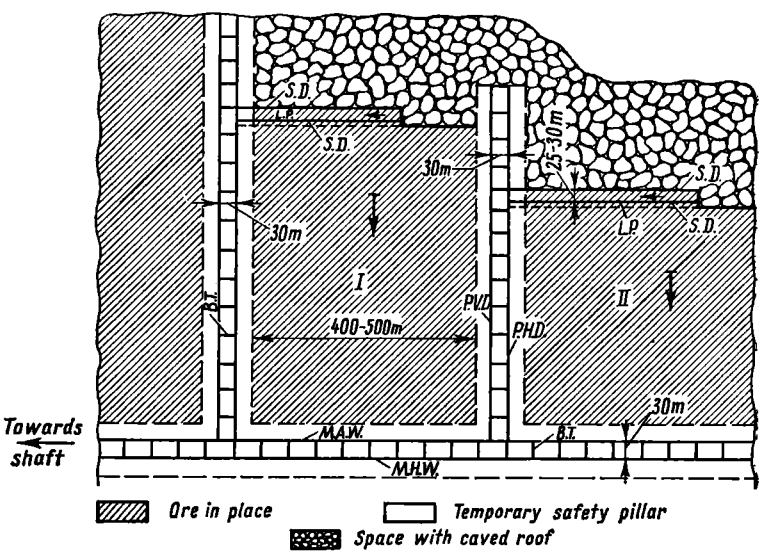


Fig. 225. Extraction of mine-field wing by long pillars:

I, II—panels; *L.P.*—long pillar;⟵ general direction of panel extraction;⟻direction of pillar recovery; *M.H.W.*—main haulageway; *M.A.W.*—main airway; *P.H.D*—panel haulage drift; *P.V.D.*—panel vent drift; *S.D.*—stope drift; *B.T.* — breathrkough. *Note*: pulkheads and air crossways are not indicated

headings being 4.9 and 7.0 sq m, respectively. All drifts are given a circular shape. With all-round utilization of multipurpose units the output per faceman may be estimated at 15-20 tons per shift. The operation of mechanized multipurpose units has been described above (see Mining of Bedded Deposits).

Advantages, Shortcomings and Technical and Economic Characteristics of Topslicing and Cover-Caving Methods

These systems are widely used in the working of ore deposits. The major advantages of the systems involving topslicing with cover caving are: the possibility of using them in difficult conditions of mining (with disturbed ore, very unstable host rocks), of robbing intervening and cap pillars, of achieving high mining intensity by stoping ore with long faces, metal support and wire-netting (mat) or flexible flooring; insignificant losses and dilution of ore when the appropriate flooring is used; lower net cost of mining than in systems with timbering and filling; the possibility of employing mechanized

Table 23

Technical and Economic Characteristics of Topslicing and Cover-Caving Methods

Mining methods	Output per faceman per shift, tons	Coef. of ore recovery	Ore dilution factor	Timber consumption, m^3/ton	Explosives consumption, kg/t
1. Topslicing with ore stoped by cuts	8-10	0.97-0.95	0.05-0.07	0.04-0.06	0.2-0.3
2. Topslicing of ore with longwalls, using metal support and mobile equipment	15-25	0.97-0.95	0.03-0.05	0.01-0.015	0.20-0.30
3. Long-pillar mining with ore stoped by twinned cuts with subsequent caving of cover rocks . .	8-12	0.90-0.95	0.05-0.07	0.03-0.07	—
4. Long-pillar mining with ore stoped by longwalls with subsequent caving of cover rocks (mechanized set)	15-20	0.93-0.95	0.03-0.05	0.005-0.01	—

Note: 1. Explosives consumption is given for medium-hard ore.
2. Losses and dilution for systems 3 and 4 do not take into account abandoned stumps.
3. Figures for pillar methods may vary considerably depending upon mining and geological conditions and mechanical equipment employed; for instance, the use of shield support may greatly increase labour efficiency and cut down timber consumption.

sets for recovering pillars in horizontal and gently sloping bedded ore deposits.

The demerits of the methods are: (1) fire hazards due to the use of wooden props and cushions; (2) caving of the ground surface and cover rocks (if it causes material damage); (3) difficulties in arranging ventilation and communication with stopes in topslicing (in the absence of stone workings); (4) full deployment of stoping operations is possible only in one level at a time.

Points 1 and 3 do not refer to stoping of ore by a longwall with the use of metal support, wire-netting cushion and working trenches. In connection with the introduction of conveyer transport, metal support and netting, flexible flooring and mobile equipment it is to be expected that the present topslicing system will undergo some changes involving a substantial increase in the size of blocks, the height of the slices extracted and the length of working faces. Important changes will also be made in the design characteristics of pillar systems in connection with the introduction of longwall stoping and the use of mechanized sets.

The technical and economic characteristics of the mining methods of Class VI are listed in Table 23.

5-8. MINING SYSTEMS WITH CAVING OF ORE AND ENCLOSING ROCKS (CLASS VII)

General

In these methods only part of the ore (first mining) needs large-scale drilling and blasting operations for its breakage, while the main bulk of it is won (second mining) with little drilling and blasting or none at all. In the latter case the ore is caved and moves by gravity and under the weight of the enclosing rocks, and then the process is called block and sublevel caving with freely caving ore and capping.

The systems under consideration find wide application in the mining of ore bodies. They are usually used independently in mining individual ore bodies, or as ancillary methods in recovering stumps and cap pillars left when employing other mining systems.

The proportional share of non-ferrous metal ores mined by systems with caving of ore and cover rocks is around 20% and that of iron ores about 50%, including 70% of iron ores produced in the Krivoy Rog basin (64% by sublevel caving and 6% by block caving). These systems are most widely used in the working of iron ore deposits, where conditions favour their application.

The general conditions for the use of methods with caving of ore and cover rocks are: unstable, readily-caving enclosing ground; the possibility of subsidence of the ground surface and enclosing country

rocks; ore of low value (these methods involve high losses and dilution of ore); no inclusions of gangue or other grades of ore (underground sorting and grading of ore are ruled out).

Class VII includes the following methods of block mining:

(a) with division of a horizon into sublevels (sublevel caving systems): with a cushion; without a cushion (extraction of ore by rooms, ore breakage by means of firing fan-shaped rounds of sectional-steel blast-holes; longholing; free sublevel caving);

(b) without division of a level into sublevels (block caving methods): free caving of ore in a block; induced caving of ore in a block with vertical or horizontal compensation chambers.

Sublevel Caving Methods

As stated above, the sublevel caving systems may be subdivided into two modifications—those with and without a mat.

Let us first consider sublevel caving with the use of a mat, which is in fact a further development of topslicing and was initially applied in the working of iron ore deposits. Sublevel caving without a mat was introduced at a later date.

Sublevel caving with a mat. In this method the level interval height is from 40 to 60 m and the level itself is broken up into blocks measuring from 20 × 20 to 50 × 50 m.

In an ore body of limited thickness the length of a block is set at up to 40-50 m. Each block is extracted by sublevels 3.5-7.5 m high. In sublevel caving with a cushion the general pattern of development and extraction of blocks is analogous to topslicing. The main distinctive feature of the method under discussion lies in the conduct of stoping operations. In topslicing, cuts are worked throughout the entire height of the productive slice, while in sublevel caving with a mat (Fig. 226) each panel is extracted in two stages.

The first stage consists in the drivage of a panel opening with a section measuring 2 × 2 to 2.5 × 2.5 m and abandonment of chain pillars of ore on its sides; the width of the pillar approximately equals that of the opening itself; also, a ceiling 2.5-5 m thick is left. The panel openings are timbered with frame-sets.

The second stage of stoping consists in retreat recovery of the abandoned chain pillars and cap pillars. The ore in the chain pillars and especially in the cap pillars is considerably disturbed by the pressure of the overlying ground; therefore stoping proceeds very rapidly with a high output per faceman and reduced cost of drilling and blasting.

Except for the point about the value of the ore, the mining and geological conditions for the use of sublevel caving with matting differ little from those for topslicing.

The method can be successfully employed in the working of ore bodies more than 6 m thick. In topslicing the cover rocks should cave immediately after removal of timbering, tightly filling the mined-out space, whereas in sublevel caving it is desirable that, following breakage of ore, the cover rocks should remain intact over a space of time needed for mucking the caved ore.

If the ore is strong and stable the minimum possible height of sublevels is adopted and a cap pillar 1-1.5 m thick abandoned. With

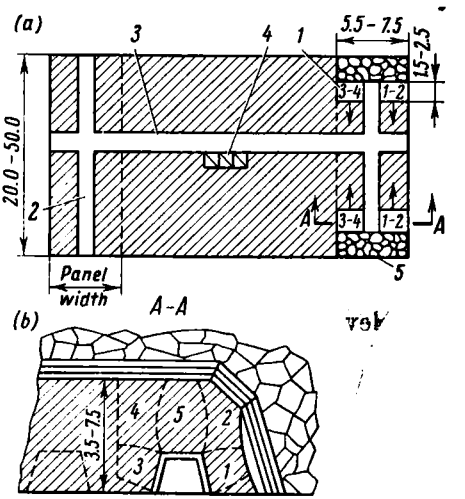


Fig. 226. Details of panel extraction in sublevel caving with matting:

a — plan view of development and subsidiary development openings in sublevel caving (*left*—first stage of mining in panel, *right*—second stage); *b*—panel section on *A-A* (numbers indicate sequence of extraction of individual parts of panel); *1*—portion of panel being mined; *2*—subsidiray panel opening; *3*—subdrift; *4*—raise; *5*—extracted and caved panel sections

an extra-high sublevel interval caving may become difficult and hazardous. The presence of soft and loose ores also restricts the sublevel interval height, since it is not always possible to regulate the extraction of the side portions of the panel opening and the caving of the cap pillar by drilling and blasting, this resulting in a non-uniform caving and in increased losses and dilution of ore. A sublevel interval of 6-7.5 m is adopted in ores of medium strength, and also in strong, but fissured ores in which caving does not present any special difficulties. The panels are from 4.5 up to 8 m wide. As the sublevel interval height increases so does the width of the panel, but in this case, too, it largely depends upon the stability of the ore. Intervening pillars of ore are left on both sides if the ore is more or less stable and if the pillar on the caving side will not be damaged during the excavation of the opening and caving of the ore (in retreat mining). Should the ore be so weak that the pillar cannot withstand the pressure and will collapse the opening should be driven near the gob area, as shown in Fig. 227, and the panel taken out by wor-

king a single intervening pillar. The width of the panel in this case should be cut down to 4.5 m. In very weak ores the panel is extracted without leaving any intervening pillars. With scraper haulage the length of the panel is set at 15-25 m.

Intervening and cap pillars of ore are caved in sections 2-3 m long. The intervening pillar on the caving side is the first to be recovered, and the cap pillar the last. Intervening pillars are caved in two, less frequently in three steps, depending upon the rigidity of the ore and

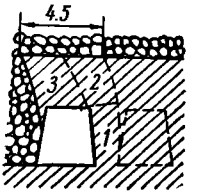


Fig. 227. Sequence of extracting individual portions of a panel in unilateral mining (indicated by numbers); mat not shown

the sublevel interval height. The sill pillar is commonly caved in a single stage because it is greatly disrupted. Before caving holes are drilled with stoper and partly with jack hammers.

When drilling the top portions of intervening pillars the back of the mined-out area is temporarily supported for safety reasons.

Operations in each successive section of a panel can be started only after the mat is laid down on the floor of the worked-out area in the preceding section and after this section is filled with the caved ground. In order to lay down a cushion of the required thickness (not less than 2-3 m) the top portion of the first level is worked out by topslicing. Only after a cushion of the required thickness has been formed can sublevel caving be begun. In sublevel caving workings of the lower sublevels serve as storage openings. In sublevel caving slushing is done in the same manner as in topslicing.

Sublevel caving with the use of mats is chiefly applied in the working of non-ferrous metal deposits containing ores of low value. The method can also be employed in the mining of pyrites and chalcopyrites, provided adequate fire-protection measures are taken and preventive silting of the gob areas carried out (the system presents a considerable fire hazard owing to the presence of timber and abandoned ore with a high sulphur content).

Sublevel caving without a mat. The system is largely employed in the working of iron ore deposits. As compared to sublevel caving with the use of mat, losses and dilution of ore acquire greater proportions. As already stated earlier, the variants of sublevel caving without a cushion now in use include: extraction of ore in rooms from special sublevel openings, a variant involving ore breakage

by means of firing fan-shaped rounds of sectional-steel blast-holes ("closed fan-shaped round"); longholing; free sublevel caving.

The longholing variant is largely preferred, its proportional application continues to increase (15.8% in 1958, 34.9% in 1961 and 53.5% in 1966). In recent years variants of sublevel caving by breakage of ore in vertical or inclined slices in tight faces by means of deep blast-holes or onto a compensation area are successfully introduced in the Krivoy Rog basin.

General Pattern of Extracting a Level and Preparation of Blocks for Stoping with Different Variants of Sublevel Caving

A level 60-80 m high is subdivided into blocks, each block into sublevels and each sublevel into panels. Blocks extend along the strike for 40-50 m, their length conforming to the thickness of the ore body. Sublevel interval heights are from 10-12 to 30-40 m, depending upon the variant of the method employed and the mode of ore breakage. In all variants of the system each sublevel is divided into panels running along or across the strike (Fig. 228). The width of a panel is between 7 and 30 m so that its ore-mining area does not exceed 500-750 sq m. Mining experience and investigations indicate that with small panels the intensity of mining in a block can be increased and rock pressure considerably reduced.

The pattern of block development should be correlated with the panel layout. Figure 229 illustrates a pattern of block preparation with the panels disposed on strike and the ore transferred down the chutes directly onto the haulage horizon. The lateral drift (*1*) and the stone raise (*7*) are driven in the foot wall beyond the ground-displacement zone. A crosscut (*2*) is pushed forward from the lateral drift (in the centre of the block) with ore-chutes (*3*) run from it.

A crosscut (*5*) and scraper subdrifts (*6*) are driven in each sublevel in conformity with the panel arrangement. Along the block borders subdrifts are connected with ventilating raises (*7*) by crosscuts (*8*). A combination service-and-vent-escape raise (*4*) runs along the contact with the hanging wall.

A variant of development involving direct slushing of ore to draw-holes has been described above. The number of draw-holes with this scheme is quite large; in order to cut it down a storage opening is made under the sublevel crosscut and the ore passed down onto it via short draw-off raises. In this case the number of main ore-chutes in a block is greatly reduced. This development pattern involves double slushing. Powerful scraper hoists operate in the storage opening.

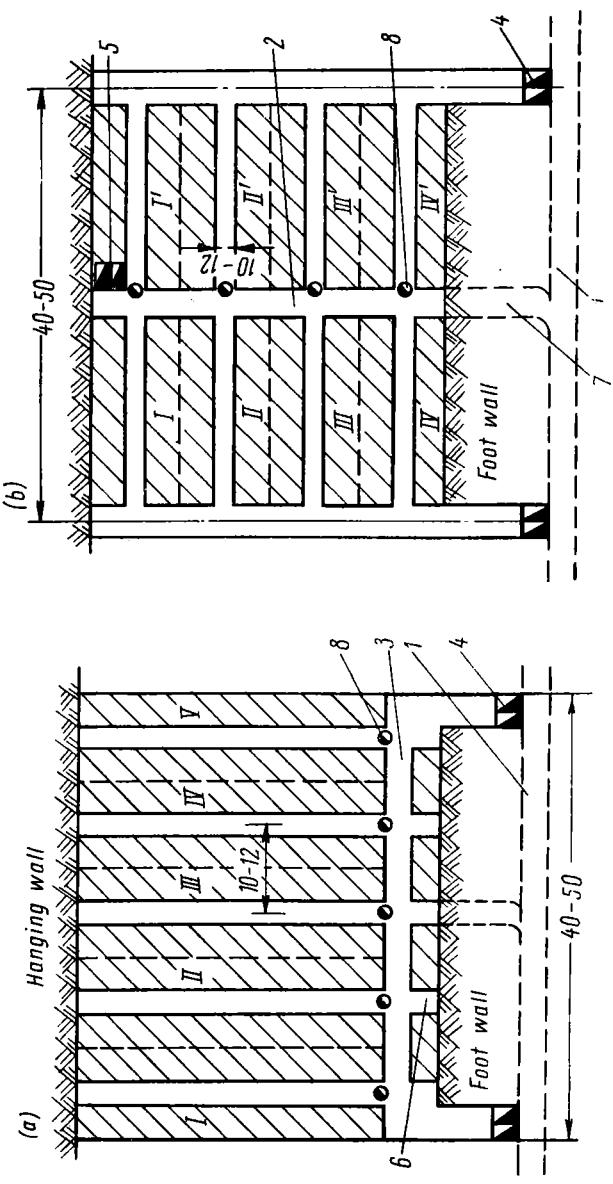


Fig. 228. Possible panel layout in the sublevel of a block (plan view):

a—across the strike; *b*—along the strike; *1*—haulage drift; *2*—sublevel crossdrift; *3*—subdrift; *4*—lateral raise; *5*—raise in ore; *6*—scraper hoist chamber; *7*—haulage level subdrift; *8*—ore-pass

Sublevel caving variant with extraction of ore in rooms from sublevel openings (Fig. 230). In this variant each panel in a sublevel is worked out by rooms up to 10-12 m long and 8 m wide, with small stumps 1.5-2.0 m thick left on all sides of the room. The room is stoped out in a manner analogous to sublevel mining (see systems of Class I). The sequence of auxiliary development work and stoping

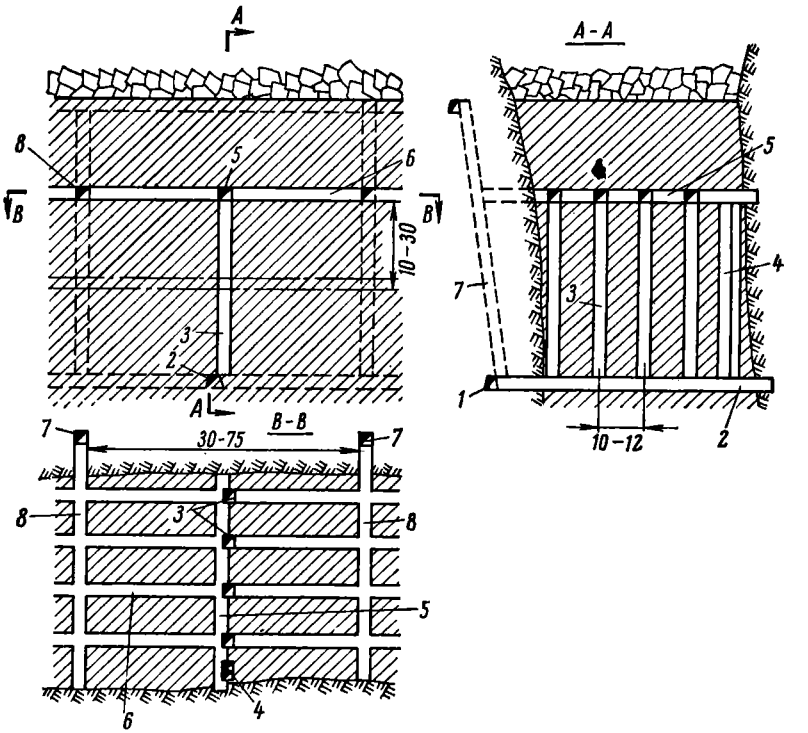


Fig. 229. Preparation of a block in sublevel caving with direct slushing of ore

operations is as follows: an undercut opening (*2*) and draw-holes (*3*) are excavated above a panel opening (*1*). After a cut-off slot (*4*) has been made and an open cut (*5*) excavated receiving funnels are formed by drilling and blasting rounds of shallow holes, this being followed by the breaking of ore from the cut by means of firing parallel sectional-steel blast-holes (*6*). The ore in the subsequent open cuts is broken in a similar fashion. From the room the ore goes through funnels and draw-holes to the panel opening, along which it is hauled to pull-holes by scrapers. After the room has been stoped out

the corresponding pillars are drilled off and blasted along the outlines of the room (*7-8*). Blasting out of the pillars results in the caving of the ceiling (*9*). During the caving of the pillars the ore mixes with the caved waste and hence cannot be recovered completely.

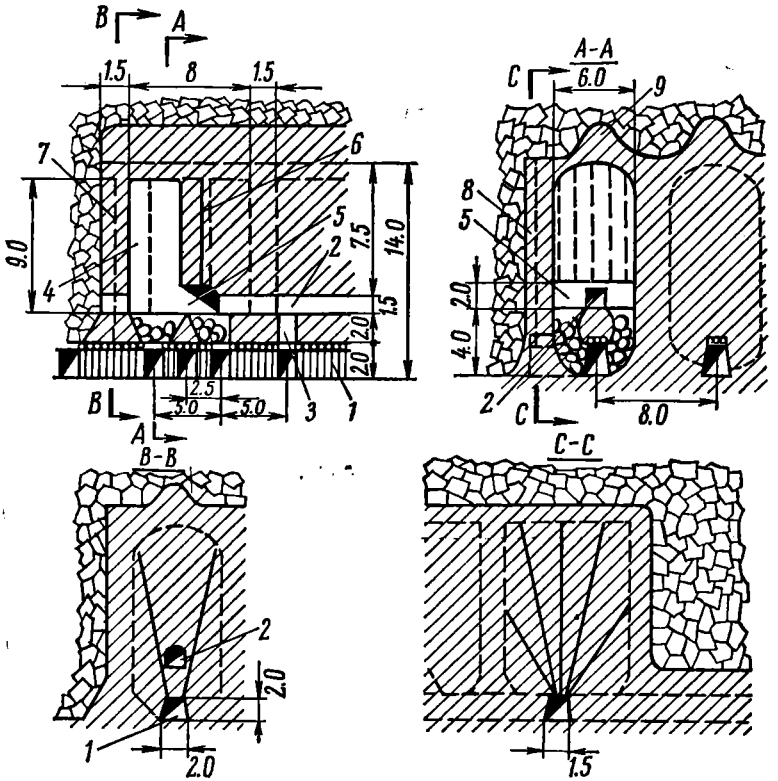


Fig. 230. Panel etrxaction in the sublevel caving variant with extraction of ore in rooms from sublevel openings

The advantages of this variant are: comfortable and safe working conditions for the drillers, high labour efficiency, and high purity of the ore obtained from the room.

Substantial shortcomings of the variant are high losses of ore and a large scope of development, especially subsidiary development work.

Variant of sublevel caving with breakage of ore by means of firing fan-shaped rounds of deep or shallow holes (Fig. 231). This variant is applied in the mines of the Krivoy Rog basin. Its main distinc-

tive feature is the sequence and mode of ore breaking. The subsidiary development workings of a panel are a panel opening, draw-holes, and a so-called undercutting chamber, 6 m long, 3 m wide and 2 m high. Draw-holes are spaced at 4 m, which is equal to the length

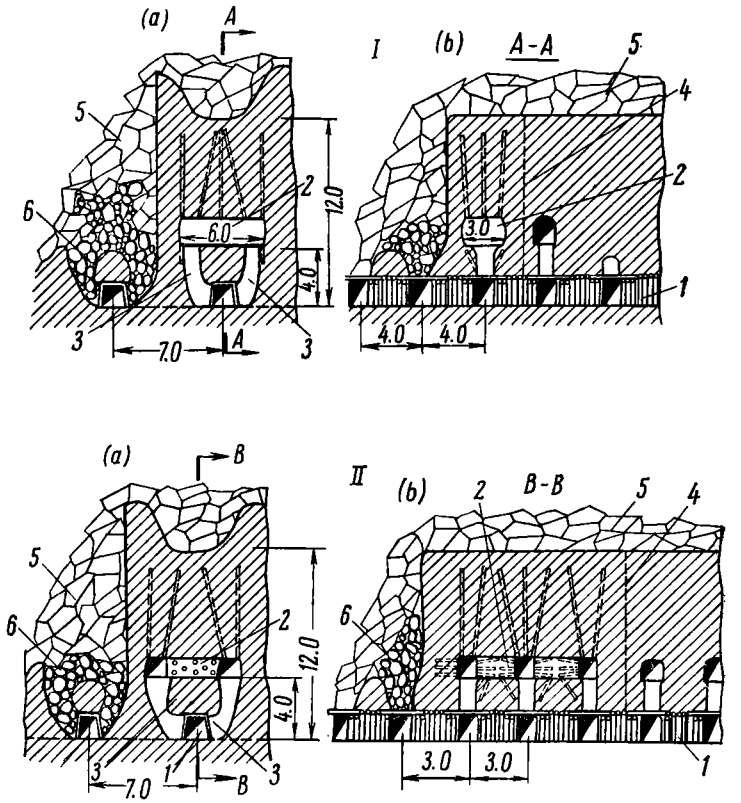


Fig. 231. Panel extraction in the variant of sublevel caving with breakage of ore by means of fan-shaped rounds of holes:

I—extraction with undercutting extending over a pair of draw-holes; *II*—extraction with undercutting extending over three pairs of draw-holes; *a*—section along minor side of panel; *b*—section along major side of panel; *1*—panel opening; *2*—undercutting chamber; *3*—draw-hole; *4*—boundary of minable portion of panel; *5*—caved rock; *6*—caved ore

of the section extracted, and arranged on both sides of the panel opening. Three rows of sectional-steel blast-holes 5-6 m deep each are drilled from the undercutting chamber to break down the ore. This is discharged onto the panel opening and hauled to an ore-chute along it. The ore can be mined simultaneously in several

panels, the panel undergoing extraction leads the next one by 2-3 sections. With a more rigid ore, panels are mined out in sections of greater length designed for three pairs of draw-holes.

The advantages of the variant are: high output per faceman and especially per driller, low consumption of materials, comfortable and safe working conditions for drillers.

The disadvantages are: high losses and dilution of ore, considerable scope of development and subsidiary development work.

Variant of sublevel caving with ore broken by longholing. This variant is actually a novel, more effective method of mining. Suffice it to say that in longholing the output per driller reaches 250-400 tons per shift, whereas with shooting sectional-steel blast-holes it is only 180-220 tons per shift.

On the basis of extensive research supported by mining practice Professor G. Malakhov, Dr. Sc. (Eng.), has confirmed the hypothesis advanced by Engineer S. Minayev stating that during the escape of caved ore via funnel openings it assumes the shape of ellipsoids (*1* in Fig. 232)[1]. All particles lying on the surface of each ellipsoid reach the discharge opening at the same time.

As the height of the ellipsoid increases so does its width (minor axis), in accordance with a certain relationship.

When the ore is drawn, only a certain part of it begins moving. This draw zone also has the shape of an ellipsoid of revolution, the so-called "loosening" ellipsoid (*2*). Its volume is directly proportional to that of the "draw" or "discharge" ellipsoid and, in conditions prevailing at the Dzerzhinsky mine, exceeds it by approximately 40 times, according to V. Kulikov. As the ore is drawn the bottom portion of the "loosening" ellipsoid gradually reaches the sphere of influence of the discharge opening (*3*) which has the shape of a paraboloid. The sphere of influence of the discharge opening is the ultimate limit beyond which particles remain stationary regardless of the amount of the material drawn from the opening. The shifting of cover rocks begins to manifest itself when the apex of the "loosening" ellipsoid passes beyond the contact plane between the ore and the overlying ground. As the discharge continues the surface of the ore contact with superjacent rocks assumes the shape of a funnel, which is called the depression cone (*4*). When this cone subsides to the level of the draw-hole the ore starts flowing together with the gangue (before this only uncontaminated ore is discharged). In this position the depression cone is called the draw cone (*5*). During subsequent drawing of ore from this opening its contamina-

[1] For more detailed information on the discharge of ore from caved blocks see G. Malakhov's fundamental work *Vypusk rudy iz obrushennykh blokov* (*Drawing of Ore from Caved Blocks*), Metallurgizdat, 1952.

tion with waste increases. The discharge of ore should cease when the content of the useful component therein drops below the stipulated standard value.

When the loosening ellipsoid reaches the ground surface a collapse funnel (*6*) forms in it.

The parameters of ellipsoids and the depression and draw cones can be determined by using V. Kulikov's formulas.

The volume of the draw ellipsoid

$$q = 0.523 \frac{h^2}{c}$$

The width of the draw ellipsoid

$$2b = \sqrt{\frac{h}{c}}$$

The sphere of influence of the discharge opening

$$x = cy^2$$

The equation for the loosening ellipsoid, the depression and draw cones is

$$y = \sqrt{\frac{x}{c}\left(1 - \frac{\sqrt{\eta q (\eta h^2 - x^2)}}{\eta c_1 q}\right)}$$

where x and y = current coordinates, m;
h = height of the ore layer, or distance to the top of the depression cone, m;
q = amount of ore drawn, cu m;
c = coefficient characterizing friability of ore (for Krivoy Rog basin it has the following values: for highly friable ore $c = 1.15$, for medium-friable ore $c = 1.35$, for ore subject to packing $c = 1.55$);
η = coefficient of secondary loosening within the loosening ellipsoid. With $c = 1.15$, 1.35, 1.55 and 1.75 it equals 1.032, 1.03, 1.03 and 1.021, respectively;
c_1 = a constant value equal to

$$c_1 = \sqrt{\frac{c}{0.523}}$$

From the foregoing it follows that in designing systems involving bulk breakage and drawing of ore one should set optimal values for the sublevel height interval and spacing between discharge openings. To minimize the volume of ore remaining between discharge openings after drawing it is necessary that the ellipsoids intersect

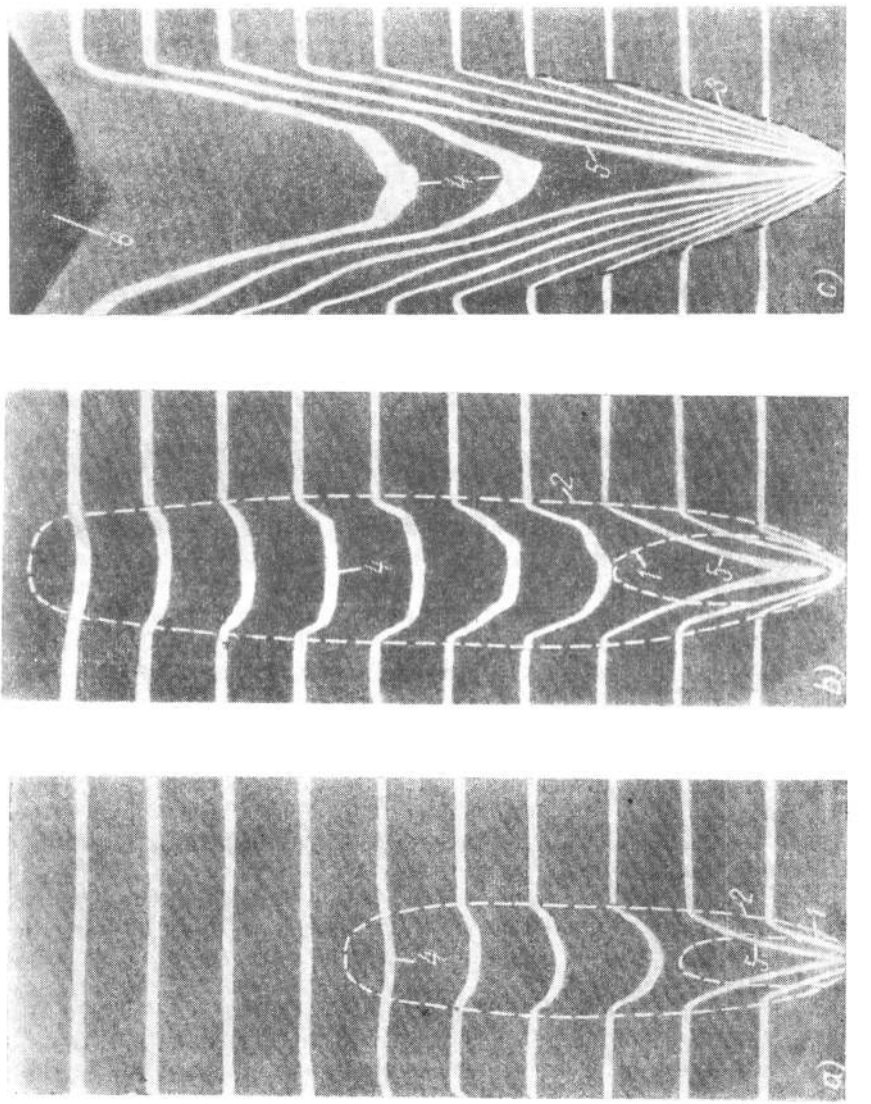


Fig. 232. Formation of discharge and "loosening" ellipsoids during escape of friable material: *a*—first stage of discharge; *b*—second stage of discharge; *c*—"outopor" of the loosening ellipsoid

one another. The diameter of the discharge opening should be at least 3-4 times the size of the largest lump of the ore to be drawn.

G. Malakhov suggests that in sublevel caving the ratio between the height h of the ore slice caved and the spacing between discharge openings l be taken as 5, and in block caving as 6, 7 or more.

V. Imenitov recommends to set the $h : l$ ratio at 5, 6 or more, whereas V. Kulikov suggests that the distance between draw-holes be determined by using the equation

$$l = \sqrt{c_2 h}$$

where c_2 is a coefficient depending upon the friability of the ore. With $c = 1.15$, 1.35 and 1.55 c_2 equals 0.7-0.8, 0.6-0.7 and 0.5-0.6, respectively.

G. Malakhov believes that under the conditions of the Krivoy Rog basin minimal losses and dilution of ore may occur if the following design characteristics of sublevel caving by longholing are adopted: sublevel height interval up to 30 m, panel width 0.7 of the sublevel height interval with one contact surface and 1.2 with two contact surfaces with the caved ground; spacing between panel draw-holes 4-4.5 m; diameter of draw-hole 1.5-1.8 m.

The more intensive the discharge of ore, the lower is the pressure on the scram level (the drawing rate per one pull-hole averages 3-4 cu m of ore per shift). As evidenced by experience the maximum reduction in losses and dilution is achieved when ore is drawn uniformly from all pull-holes over the panel area with a horizontal contact between the surface of the broken ore and the caved superincumbent rocks.

The ore from sublevel panels or from a block (in block caving) should be drawn according to a schedule prepared under the supervision of a technical inspector.

The basic points of the ore-drawing theory, as stated above with reference to sublevel caving, are also applicable to block caving, which is discussed below.

The sublevel longhole caving method is widely used in the Krivoy Rog basin. There are good reasons to believe that this system will be one of the principal methods in the mining of Soviet ore deposits, along with block caving.

The general pattern of sublevel caving with the ore broken by blasting long horizontal holes is presented in Fig. 233.

Below we give a brief description of this method as applied in the Krivoy Rog basin (Fig. 233). Like in all systems of sublevel caving the ore in a horizon is taken out in blocks. The development workings in a block are haulage openings (drift, crosscut), a stone raise, and sublevel crosscuts.

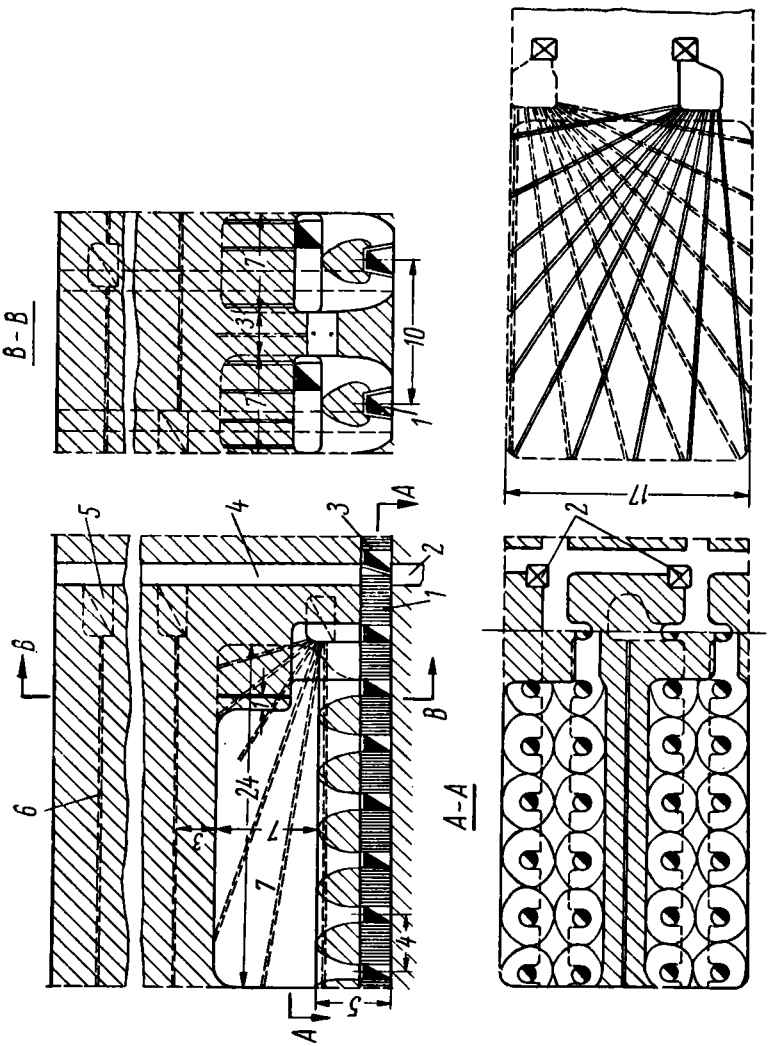


Fig. 233. Sublevel caving of ore by longholing:
1—subdrift; 2—ore-pass; 3—sublevel crossdrift; 4—drill raise; 5—drill chambers; 6—long holes; 7—compensation chamber

Each sublevel with an interval of 15-30 m is worked out by individual panels up to 25 m long (in the diagram the panels are disposed on strike). In unstable ores the panel size may be reduced. Prior to actual mining subordinate development openings are excavated in each panel (two subdrifts, two short raises with long-hole drill chambers, draw-holes with receiving funnels and an undercut opening). Undercutting is done with allowance for an increase in the ore volume due to its loosening (when all slices in a sublevel are blasted out simultaneously). Pull-holes are commonly spaced at

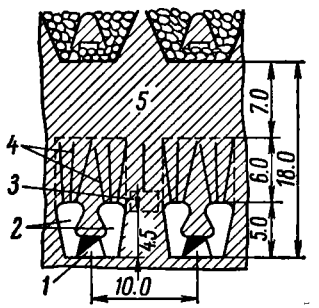


Fig. 234. Cross-section of a panel undercut with sectional-drill steel holes:

1—panel slusher opening; *2*—draw-hole; *3*—opening in a temporary pillar between undercutting chambers; *4*—long holes drilled to form undercutting chamber; *5*—solid ore in place blasted out after undercutting it with two fan-shaped rounds of deep horizontal holes

4 m. As the undercutting of a panel proceeds, long holes are perforated from drill chambers. The holes are arranged fan-wise. Long holes in each slice of a panel are run in different directions (Fig. 233), their number depending on the panel size and the hardness of the ore. The holes are drilled with a rotary rig, they are 85 mm in diameter and from 10 to 25 m long. In the Krivoy Rog basin the average rate of advance per rig is 10-15 m per shift. The overall length of long holes in a slice reaches 200-250 m. Krivoy Rog mines yield 20-40 tons of ore per metre of hole. Long holes are charged with special blasting cartridges of increased diameter, up to 500 mm long. Simultaneous long-hole blasting in all slices of a panel is done with a detonating fuse.

Two to three hundred grams of explosives is consumed per 1 ton of broken ore. The broken ore is discharged onto panel workings and then slushed to ore-chutes along them. Then it goes to a storage crossdrift or directly to the haulage horizon.

The advantages of this variant are: high labour efficiency and mining intensity, low consumption of materials. Losses and dilution of ore can be appreciably reduced by adopting correct design characteristics of the system.

The efficiency of the method can be raised by undercutting the panel with sectional-drill steel holes (Fig. 234), drivage of trenches or blasting the ore from an undercut opening with fan-shaped rounds of blast-holes. These measures enable the efficiency of a driller in

undercutting to be increased to 100-120 tons per shift as against 30-40 tons with short-hole undercutting.

In making a general assessment of the above-discussed method of sublevel caving it should be admitted that breakage of ore with horizontal long holes entails the excavation of many temporary

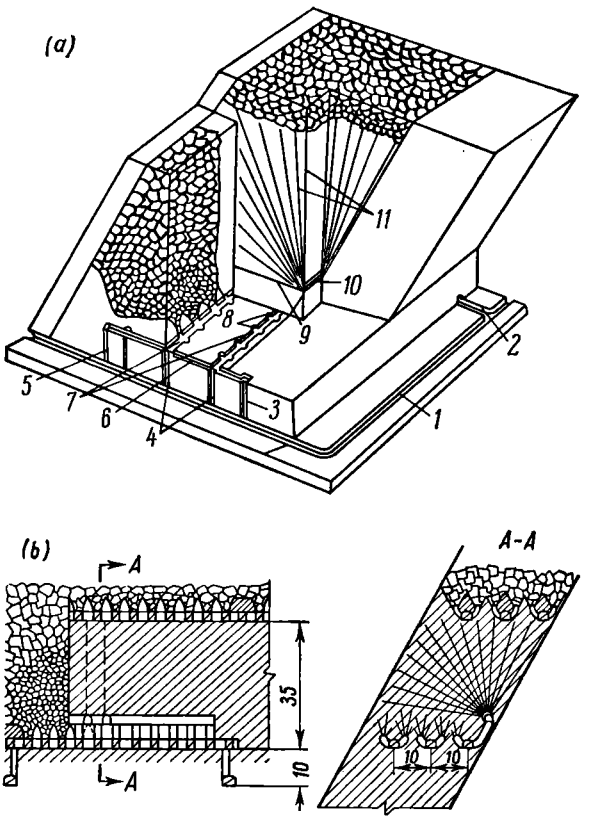


Fig. 235. Sublevel caving by vertical slicing in tight faces:
a—general view; *b*—sections along and across the strike; *1*—lateral haulage drift; *2*—haulage subdrift; *3*—manway; *4*—ore-chute; *5*—passageway; *6*—connecting crosscut; *7*—scram drifts; *8*—draw-holes; *9*—undercutting level; *10*—undercutting drift; *11*—long holes

development openings. In connection with the progress in the technique of drilling (the possibility of making holes with drill units in any direction) sublevel caving can be done more efficiently by breaking ore with upward fan-shaped rounds of long holes into compensation chambers or in tight faces, if the mining and geological conditions permit it (Fig. 235). The use of upward fan-shaped

rounds of long holes in the breaking of ore enables the volume of subsidiary development work to be considerably reduced.

Drilling off of solid ore in place with long up-holes is done from drill cutouts located in scraper drifts at the sites of draw-hole recesses. The main fan-shaped round of long holes should be run from a drill cutout located in the foot wall above the subdrift. The distance between rounds of long holes on strike should equal the spacing between draw-holes. Breakage and drawing of ore are accomplished in vertical slices 9-12 m thick.

Breakage of ore in tight faces, when the mining and geological conditions permit it (marked jointing of hard and stable ores in a direction favouring their breakage) and with the appropriate parameters of the method enables the volume of temporary development workings to be kept to a minimum, helps improve fragmentation of ore and greatly increase the intensity of drawing ore from a panel by using suitable transport facilities (powerful scraper hoists for short hauls, mobile loading-and-transport units, conveyers and hydraulic transport).

In some instances pilot operations with breaking of ore in tight faces produced negative results[1], while in others the results were positive (at the Maslyansk mine, where about 50% of the aggregate ore tonnage is mined by breakage in tight faces). On the basis of laboratory investigations and field observations V. Imenitov, Dr. Sc. (Eng.), gave a favourable account of ore breakage in tight faces as practised in the Maslyansk deposit. His conclusion is that there must exist some interrelated critical values (for given conditions) of the minimal diameter of blast-holes, the least unit consumption of explosives, and the maximal thickness of the slice to be blasted. Investigations on breakage of ore in tight faces should be continued.

Bulk breakage of ore in tight faces under suitable mining and geological conditions will certainly help improve methods of sublevel caving and develop their new effective variants.

In analysing the investigations into the system of sublevel caving with longholing carried out for many years in the Krivoy Rog basin research and field workers found it expedient to adopt the method of working out blocks starting from the centre of the occurrence towards its flanks and from the foot wall to the hanging wall. This should be done in panels measuring 25 × 30 m with the ore longholed

[1] At the Dzerzhinsky mine in the Krivoy Rog basin blasting in tight faces with insufficiently stable ores resulted in considerable disturbances of the solid ore and of openings driven therein and, consequently, in high expenses for restoring them. Since the adjacent caved ground was not compacted the drawing of ore in narrow strips of considerable height led to appreciable dilution and high losses of ore.

and blasted out into compensation chambers, or in tight faces. The extensive introduction of this technology of stoping in 1959-1961 helped to minimize rock pressure, cut down ore losses and bring the output per miner at underground metalliferous enterprises of the USSR to 17 tons per shift.

Variant of spontaneous sublevel caving (Fig. 236). This variant is employed in working individual deposits distinguished by difficult geological and mining conditions, i.e. by instability of the ore and high rock pressure. A sublevel is allowed to cave spontaneously in separate panels with an area of 300-600 sq m, following its undercutting by shooting sectional-steel holes drilled from pull-holes on the scraper level. A block is divided into two flanks shifted with respect to one another by one half of the sublevel interval height along the vertical plane. Each flank is served by a separate sublevel crosscut, which is not driven along the block axis, but pierces the solid ore of the other flank, thus obviating the necessity of leaving sill pillars which accelerate the collapse of mine workings. Recovery of such pillars is a labour-consuming operation attended by high losses of ore.

In spontaneous sublevel caving labour productivity per faceman is very high, reaching 55-60 tons per shift.

Among the modifications of sublevel caving applied in foreign metal mines mention should be made of a variant involving the drilling of sublevels with fan-shaped rounds of blast-holes directly from sublevel drifts or crosscuts with subsequent blasting and loading of the caved ore into self-propelled mine-cars by mobile loaders. In this variant mining operations proceed in the following sequence (Fig. 237*a*): from a lateral drift (*1*) ore-chutes (*2*) are put up in the foot wall every 50-60 m (at a Swedish mine groups of chutes are raised, each consisting of 5-6 chutes spaced at 200-250 m). A service raise (*3*) is driven up near the ore-chute. Between main horizons inclines are run at an angle of 10° every 400-500 m to facilitate flitting of self-propelled equipment from one horizon to another and into individual sublevels. In each sublevel up to 10 m high subdrifts (*4*) are driven which are connected with the ore-chute and service raise by crosscuts (*5*). Sublevel crosscuts (*6*) are pushed forward every 6-7 m from the subdrift until they reach the hanging wall. The ore is broken by drilling fan-shaped rounds of long holes from the hanging to the foot wall (Fig. 237*b*). Holes 32-38 mm in diameter are drilled with perforators mounted on self-propelled mine jumboes (the drilling efficiency of such a unit reaches 150 m/shift). After a round of holes has been blasted the ore is loaded by means of a crawler-mounted machine of the "Joy" type (equipped with gathering arms) into self-propelled 12-15 ton mine-cars (Fig. 237*c*). The loading and haulage efficiency of one such machine with two self-

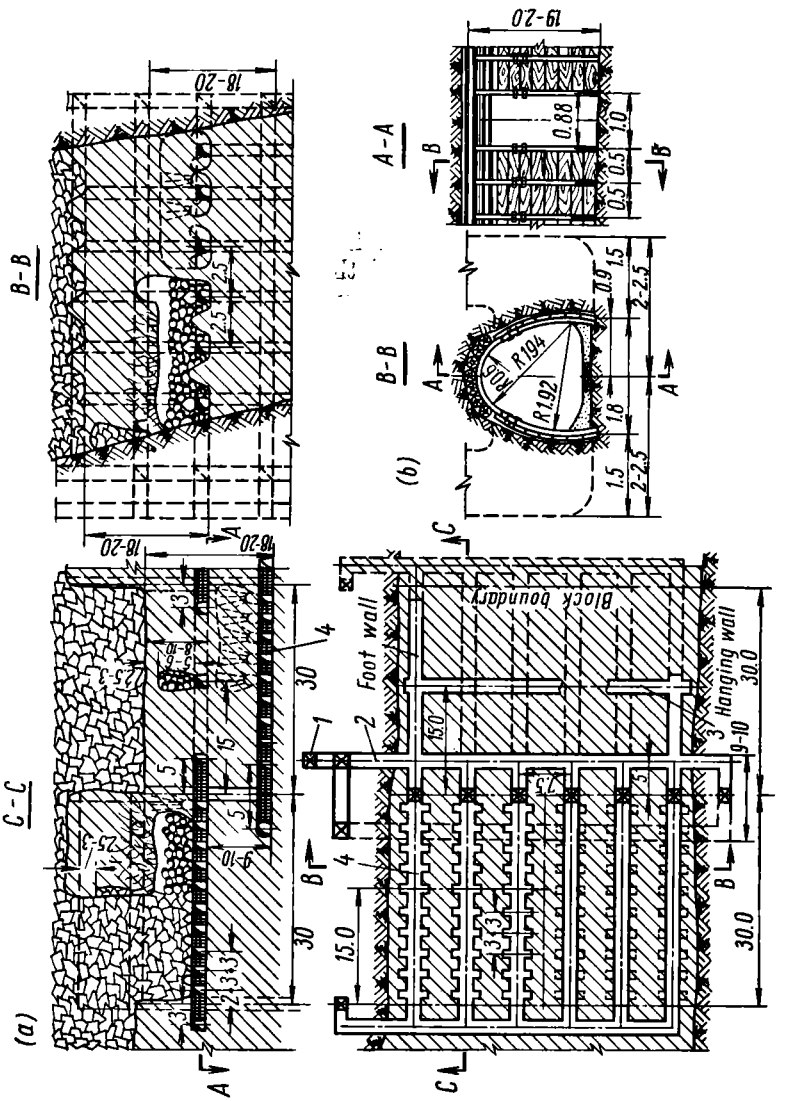


Fig. 236. Spontaneous sublevel caving:
a—general layout; *b*—timbering in scram drifts with high rock pressure: *1*—air-raise; *2*—sublevel crosscut; *3*—fringe crosedrift; *4*—scraper drift

propelled mine-cars attached to it reaches 1,000 tons of ore per shift. This modification finds wide application in Swedish mines with stable ore (without the timbering of sublevel workings). In this variant the scope of subsidiary development work is not great owing to the absence of draw-holes and receiving funnels. Output per man reaches 80-85 tons per shift at faces and 20-30 tons per man for the whole mine.

The technical achievements of Swedish mines in recent years include an increase in the sublevel interval height (up to 20 m in

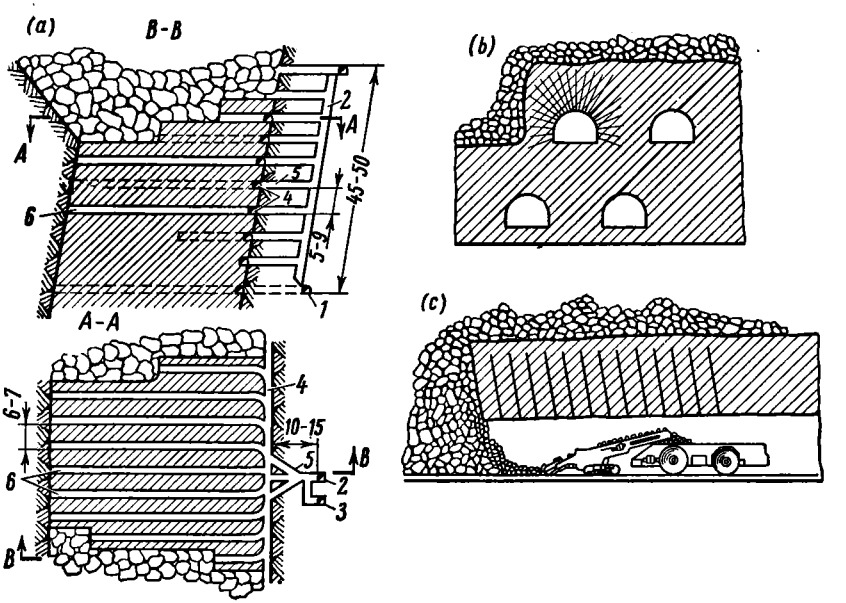


Fig. 237. Swedish variant of sublevel caving

pilot blocks) and adoption of rational design features of sublevel stopes. Swedish engineers I. Janelid and R. Kvapil recommend a formula for approximate calculation of some geometrical parameters in planning the sublevel caving method, based on the size of ore particles from the largest lumps (600 mm) to powder (Fig. 238).

The sublevel hade angle α_1 may be taken as 90°, less or more. The $S_o : S_r$ ratio, i.e. the ratio of the ore lump size to that of barren rock, is very important. When $S_o : S_r < 1$ the sublevel hade angle should be less than 90°, when $S_o : S_r > 1$ it should be more than 90°. The effect of other factors may be formulated as follows: the smaller the depth of the loading machine penetration and the greater the friction at the sublevel boundary, its height and the ore-drawing

Parameters	*Formulas for approximate calculation*
1. Sublevel interval height S, m . .	$9.0 \leqslant S \leqslant 20$
2. Width of pocket v, m	$v = 0.2S$
3. Optimal width of sublevel drift or crosscut B, m	$B = 10D$ (where D = diameter of the largest lump)
4. Optimal width of an intervening pillar between drifts or crosscuts P, m	$P = S - B$
5. Optimal distance of haul x, m	$x = 0.6H \leqslant v$ (where H = height of sublevel opening)

rate, the more positive must be the sublevel hade angle. The more positive the sublevel hade angle, the smaller the required width of the cut and of the ore flow. The values of these parameters are determined more exactly by experience. The technical and economic characteristics of the sublevel caving method at one of the big Swedish mines are: sublevel interval height 9-13.5 m, sublevel opening section 5.5 × 3.7 m, diameter of long hole 51 mm; Atlas Copko drill rig, model Simba 22; drilling efficiency 250 metres of hole or 2,600 tons per

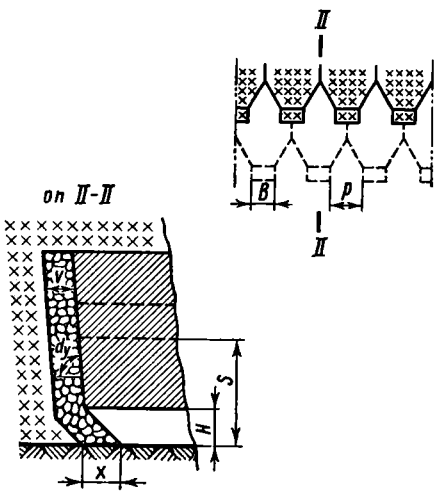


Fig. 238. Sublevel caving in thick ore bodies with a crosscut driven across the strike

shift; loading machine type Joy 18HR; self-propelled 7-11.7 cu m mine-car; average haul distance 150 m; loading efficiency 1,200 tons per shift, or 400 tons per man per shift; output per underground miner 55 tons per shift.

The USSR State Projecting Institute of the Non-Ferrous Industry has devised a new variant of sublevel caving with the use of self-propelled loading, transport and drilling equipment (Fig. 239). The distinguishing feature of this modification is the drivage of special drill openings in each sublevel to ensure better piercing of a sublevel with fan-shaped rounds of long holes. The sublevels are worked out along the strike.

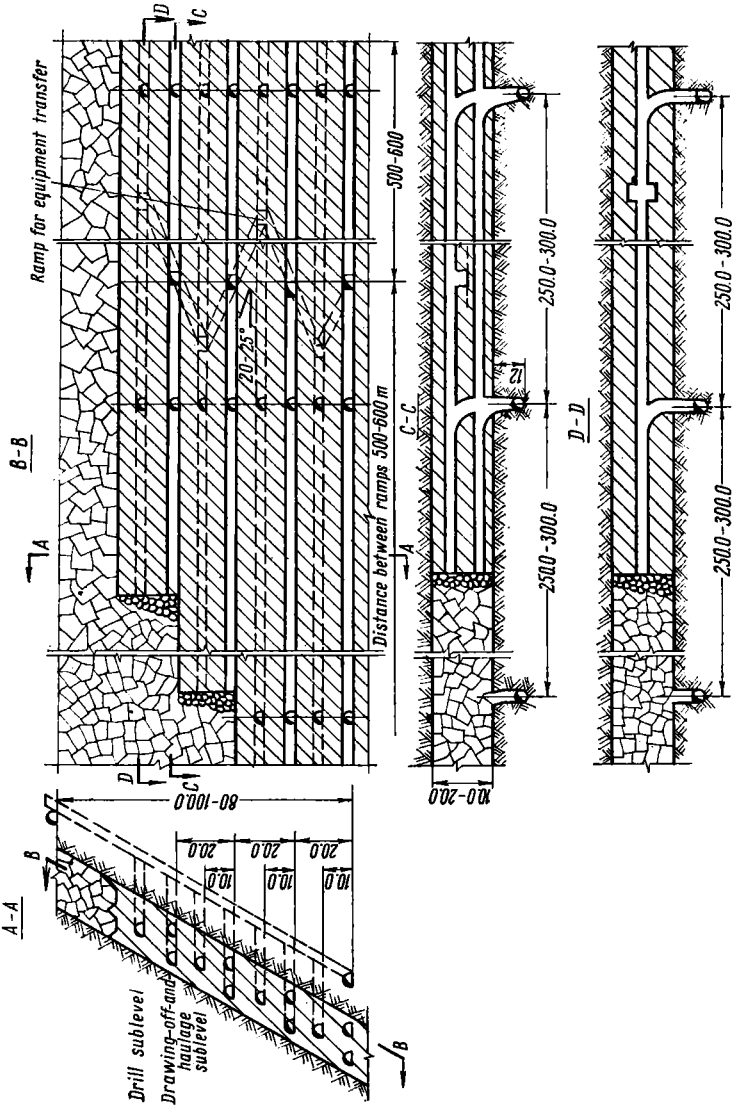


Fig. 239. Sublevel caving with the use of self-propelled flexible equipment

Block Caving with Spontaneous Caving of Ore

Essence of the system (Fig. 240). A horizon with an interval height from 40-60 to 80-100 m is broken up into blocks each with an area of from 30 × 30 to 50 × 100 m. At the base of the block a series of openings are excavated on the haulage, scraper (or grizzly) and undercut horizons. Undercut workings are driven with abandonment of provisional horizontal supports (pillars). At the same time a series of marginal horizontal openings are driven in the vertical plane along the block boundaries, which weaken the cohesion between the block and the adjacent solid ore in place. The marginal openings are sometimes replaced by a vertical slot along the block perimeter (from the base to half the block height).

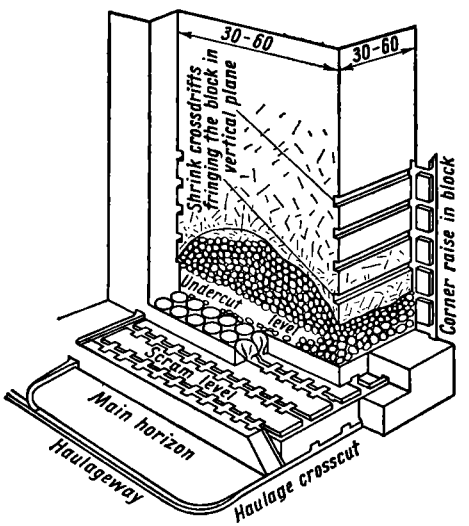


Fig. 240. Spontaneous block caving of ore

Following artificial destruction of supports at the base of the block, accomplished by longholing, the ore caves in spontaneously and gradually fills the undercut workings. As the ore is drawn from the bottom of the block onto the grizzly or scraper horizon the spontaneous caving of the ore extends to the top portions of the block. The collapsing ore breaks into separate pieces under its own weight and that of the overlying ground.

The bulk of the undercut ore in a block (up to 90-93% of its total mass in the block) is mined without drilling and blasting and without expenditure of explosives and mine timber.

The advantages of the block caving method consist in very high labour productivity and low consumption of materials. The drawbacks are high losses and dilution of ore during its discharge. The economic damage due to losses and dilution diminishes if host rocks contain appreciable amounts of useful components. Then it is possible to increase the total amount of ore drawn from the block.

Leaner ore, the last to be pulled out, is sent to a concentrating plant.

In the USSR block caving was first introduced back in 1948-1950 in the Krivoy Rog basin, in 1950-1956 it was extended to a number of mining enterprises of that region and some mines of the non-ferrous metal industry, but its proportional use has now decreased because of the rigid requirements imposed by it.

In 1961 this system accounted for only 2.4% of the total ore mined in the Krivoy Rog basin, since it had largely been supplanted by effective variants of induced sublevel and block caving methods whose application is not restricted by rigid requirements.

Conditions for the use of block caving with spontaneous caving of ore. Apart from the general conditions for the application of methods involving the caving of ore and enclosing rocks, spontaneous block caving requires the presence of the following mining and geological conditions: (1) ore body thickness of at least 25-30 m, since otherwise the scope of development and subordinate development work per ton of ore mined increases while losses and dilution reach an excessively high level; (2) tendency of ore to cave spontaneously in small lumps or to become fragmented after caving under the pressure of the overlying ore and ground. This major condition can be met only if the ore possesses very definite physical properties, and it imposes rigid limitations on the use of the system. Deposits worked by spontaneous block caving are characterized by intensive jointing of the ore, its lamination, or by the presence in the solid ore of intercalations of soft minerals (geologists should focus their attention on this feature during prospecting for iron ores and lean ores of non-ferrous metals); (3) ore of low value (since the high losses and dilution of ore typical of this system cause considerable economic damage in the mining of valuable ores); (4) the presence of useful components in host rocks which reduces the economic losses owing to the contamination of the ore with enclosing country rock; (5) low sulphur content in the ore, for the presence of large amounts of sulphur may cause self-ignition when individual pieces of the caved ore rub against each other; (6) no tendency of the ore towards packing or rapid oxidation.

Among the other conditions for the successful application of spontaneous block caving we will note the necessity of proper dressing of diluted ore and enhancing the intensity of ore discharge. The concentration of diluted ore will increase the total amount of the drawn mineral and consequently reduce losses of the metal.

In the Krivoy Rog basin the average rate of ore discharge at the time of spontaneous caving was comparatively low, 0.3-0.5 tons per day from 1 sq m of the block area, which resulted in a considerable pressure upon scraper level workings.

To cut down the rock pressure the size of a block on strike should be kept down to 40-50 m. The blocks within the bounds of the mine-field should be worked from the centre to the flanks of the deposit.

To raise the intensity of ore drawing one should also use effective haulage facilities, such as powerful scraper hoists, conveyers, and mobile equipment.

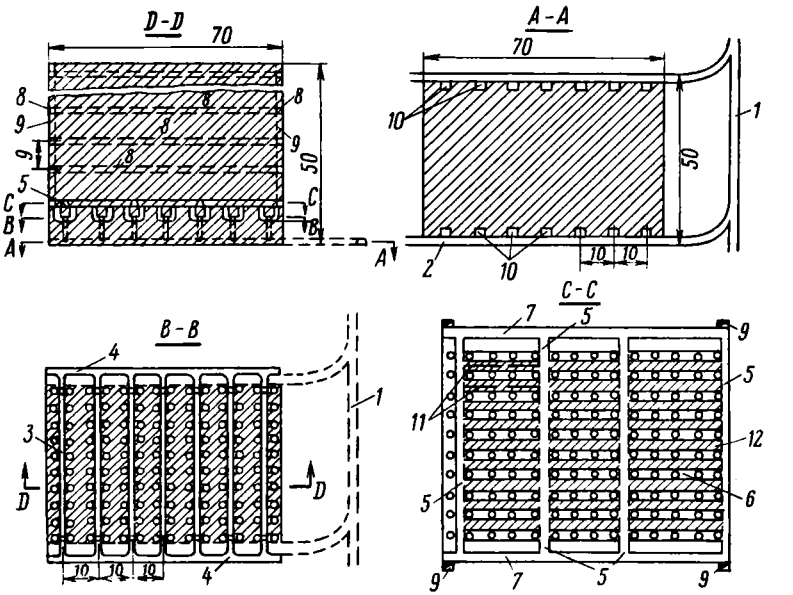


Fig. 241. Layout of mine openings in a block worked by spontaneous caving (cutoff mining):

1—lateral drift; *2*—haulage crosscut; *3*—scraper drift; *4*—connecting crossdrift on scraper level; *5*—drift connecting undercut crosscuts; *6*—opening connecting draw-holes; *7*—connecting crossdrift on undercut level; *8*—fringe opening at block boundary; *9*—raise used for drivage of fringe openings; *10*—ore-chute; *11*—long holes drilled from a drift to break down supporting ore pillars in undercut level; *12*—temporary pillar

Spontaneous block caving in the working of Soviet ore deposits. We will now briefly outline the mining techniques with spontaneous block caving and a scraper horizon as applied in one of the blocks at the Dzerzhinsky mine (the Krivoy Rog basin). Simplified schematic representations of the horizons and an elevation of the block are shown in Fig. 241.

The mining area where cut-off mining with block caving was employed consists of seven deposits with ores having different physical and mechanical properties, their hardness factor ranging from 3 to 6 in M. Protodyakonov's scale. The hanging wall of the bed

contains hydrohematite hornfels (35% Fe) occurring in a layer up to 5 m thick with a hardness factor $f = 6$-8. The hornfels are in contact with clay-sericitic schists, whose hardness factor $f = 2$-3. The foot wall consists of hornfels with $f = 8$ containing up to 37% Fe. The block measured $70 \times 50 = 3{,}500$ sq m with a level interval of 51 m.

In the foot wall of the deposit a roadway and two crosscuts were driven in the haulage horizon. Starting from crosscuts short ore-chutes were put up every 10 m to the scraper level lying 7 m above the haulage horizon.

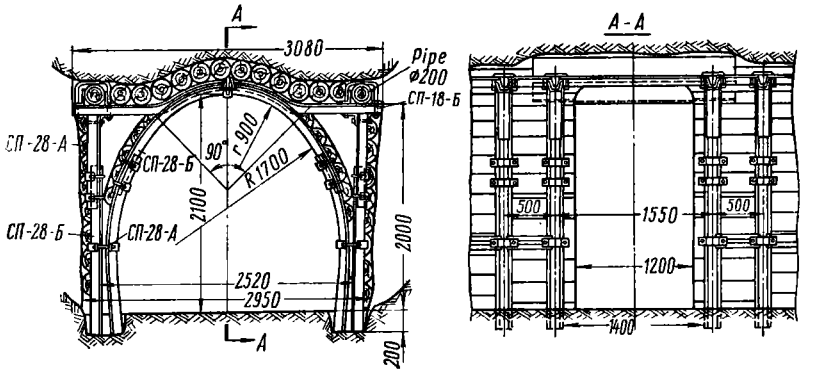


Fig. 242. Timbering at the junction of a draw-hole and a scram drift

In the scraper level 2.2×2.2 m companion headings are advanced every 10 m which are connected by 2×2 m ventilation crosscuts near the block boundaries. The drivage of the scraper level 7 m above the haulage level ensured the stability of the hauling roadways and enabled the ore to be stored in chutes so as to reduce the dependence of the underground transport on the drawing of ore. Pull-holes spaced at 5 m were raised on both sides from the scraper level drifts to the undercutting horizon. The scraper level drifts were timbered with three-piece frame-sets; at the sites where drifts joined pull-holes metal square-sets were installed. In case of increased pressure, junctions of scraper openings and draw-holes were provided with yielding metal support (Fig. 242). Where necessary the discharge openings of draw-holes were covered with timber slabs inserted into the grooves of the timber pieces. Rails were laid on the floor of scram drifts, and wooden timbering posts had strip iron lagging at the bottom.

To separate the vertical walls of the block from the solid ore in place cut-out raises were put up at the block corners; in addition

sublevel marginal crosscuts and drifts measuring 1.8 × 2.2 m and spaced at 8-10 m were driven vertically. Special crosscuts connected by drifts along the draw-hole lines were excavated in the undercutting horizon to undercut the block. The pillars formed as a result of the drivage of these openings were pierced from the drifts with rows of horizontal blast-holes, each 9-10 m long. After the pillars had been blasted the ore was drawn in accordance with a special plan (commencing from the foot wall). In the system under consideration correct and uniform drawing of ore from the block is one of the most important means to reduce losses and dilution of ore.

During the development and cutting up of the block an aggregate of 781 m development and 4,278 m temporary openings were driven. The percentage of reserves, according to the types of operations. with respect to the total ore reserves in the block was:

In the bottom—between the haulage and undercutting horizons	21.4
From development workings	0.7
From temporary workings	5.3
From actual stoping (drawing of the caved ore)	72.6

(including undercutting 3.8%, funnels 0.8%, spontaneously caving mass of solid ore 68%). The bottom is taken out by using the same system as in mining the subjacent level.

Induced Block Caving

The general pattern of this system is presented in Fig. 243. This method differs from spontaneous block caving in that the ore in a block caves, after undercutting, not under the weight of the ore and of the collapsing cover ground, but as a result of firing charges in deep blast-holes. Long-hole rows in a block are spaced at a certain distance (multiple blasting of all rows of long holes is done at short time-intervals).

Experimental introduction of induced block caving was started in the Krivoy Rog basin before 1941, and since 1945 the method has been successfully employed in a number of iron and non-ferrous metal mines. As far as its technical and economic characteristics are concerned the system is considered to be very effective, and its field of application is wider as compared with spontaneous block caving. With induced block caving it is possible to achieve uniform fragmentation of ore (which cuts down expenditures on secondary breaking) and to improve the conditions of the discharge of ore.

Conditions for the use of induced block caving. This system can be used not only with ores subject to spontaneous caving in small fragments, but also with hard ores which do not exhibit this property, with ore bodies having less regular outlines (the boundaries of the caving zone can be fixed by the corresponding disposition of long holes), and with host rocks of low mineral content (controlled caving enables dilution of ore with enclosing barren rocks to be cut down).

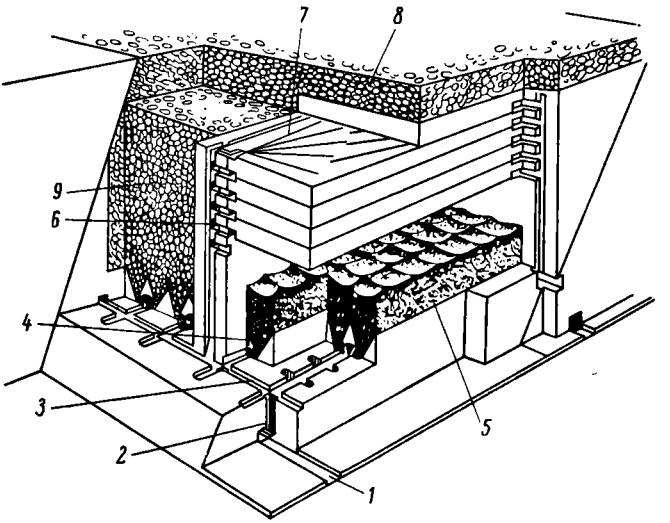


Fig. 243. Block caving with ore broken by longholing:
1—haulage level; *2*—ore-chute; *3*—scraper level; *4*—pull-holes; *5*—undercut level; *6*—drill cutouts; *7*—long holes; *8*—rock capping; *9*—ore

The other conditions for the use of induced block caving are about the same as for spontaneous block caving.

Induced block caving is used either with ore breakage into horizontal or vertical compensation chambers or without compensation chambers.

Below we give a brief description of a typical variant of induced block caving into horizontal compensation chambers as practised in iron ore deposits of the Krivoy Rog basin.

Mining with induced block caving into horizontal compensation chambers (Fig. 244). Blocks in a level are taken out for a length of 40-50 m on strike, their width being selected in accordance with the thickness of the ore body (in areas exposed to high rock pressure extraction of blocks of reduced size with an area of 800-1,200 sq m and a width of 20-30 m has been commonly used in the Krivoy Rog basin in the last few years). In a block of smaller size it is easier

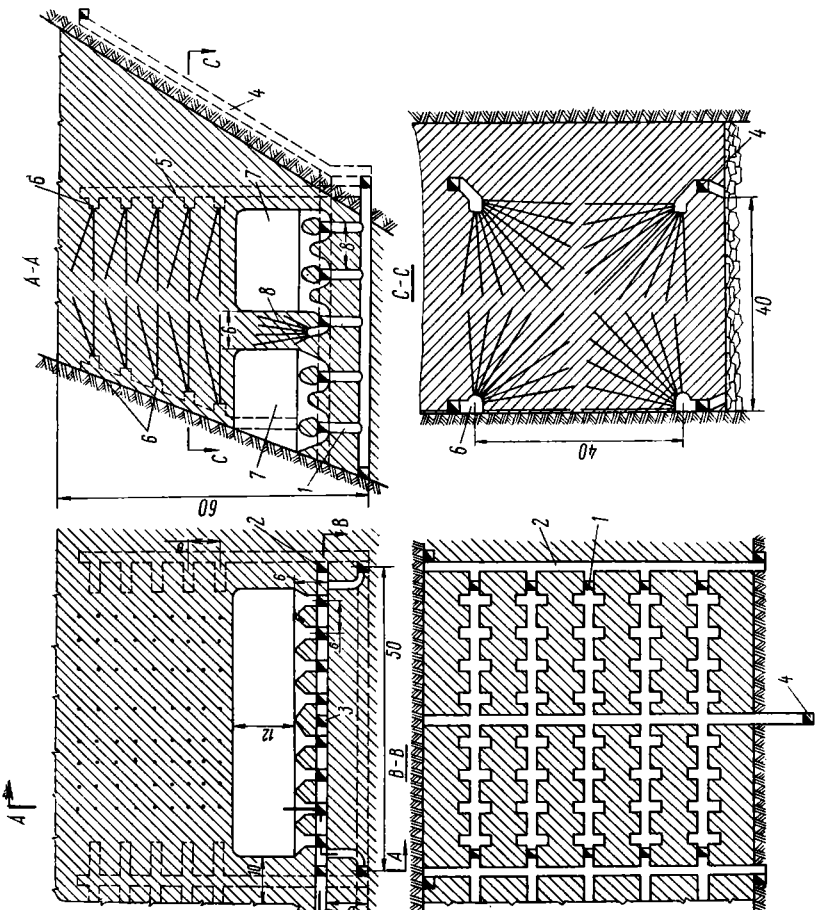


Fig. 244. Block caving induced by longholing

to ensure high mining intensity and thus reduce the rock pressure. A block is prepared for stoping by driving a set of workings on the haulage, scraper and undercut horizons. A ventilation crosscut (*3*) and a stone raise (*4*) are made for proper aeration. To pierce the solid mass of ore in a block, drill raises (*5*) with drill chambers (*6*) are excavated. The ore slides along ore-passes (*1*) from the scram level down to the haulage horizon. Communication is maintained via connecting crossdrifts (*2*). Stoping is initiated by the excavation of two horizontal compensation chambers (*7*) between which a temporary intervening pillar (*8*) is left. The area of the undercutting chamber is determined in accordance with the stability of the ore in the block (in the Krivoy Rog basin it varies between 300 and 1,000 sq m), the chamber height being chosen so as to provide the necessary compensation for the increase in the volume of the ore on loosening. Compensation chambers are made by taking out the ore by sublevel caving (variant "room over draw-holes" or "closed fan"). The block is broken in slices 6-7 m thick by shooting fan-shaped rounds of deep holes. The temporary pillar is blasted first, the solid ore in the block coming next, being blasted out in slices with delay intervals of 1-2 sec. The ore is drawn off in the same manner as in spontaneous block caving. A block of reduced size is drilled from two raises. In the Krivoy Rog basin explosives consumption with this method is 0.20-0.30 kg per ton of ore. The best results with this system have been achieved in the shaft "Gigant" of the Dzerzhinsky mine:

Amount of development and subsidiary development work per 1,000 tons, m	1.9
Output per faceman, tons/shift	60.6
Losses, % %	about 15
Dilution, % %	about 10

Induced block caving into vertical compensation chambers (Fig. 245). The system is commonly employed in working hard ores with tectonic dislocations in the solid mineral, where broad open chambers may cause premature caving of the ore mass, especially of the ceiling. This modification is also used when it is impossible or inexpedient to drill horizontal blast-holes (horizontal bedding or jointing). Due to the considerable successes achieved in drilling long upward holes the system holds good promise for large-scale application.

The ore in a level with an interval height of 30-50 m is won block-wise. Prior to stoping, a series of mine openings are driven on the haulage, scraper and undercut horizons, and also a number of ancillary openings are made. Stoping is initiated with the excavation

of vertical compensation chambers by firing deep downward blast-holes made from a special drill floor in the top portion of the block. Compensation chambers are made 6-8 m wide to offset the increase in the volume of the ore on loosening. The excavation of a compensation chamber is commenced with an opening slot by driving up a cut-out raise and then enlarging it by firing a round of long vertical blast-holes. The subsequent shooting of long holes with drawing of the ore results in the formation of an open compensation chamber. Once the compensation chambers are completed the ore is broken

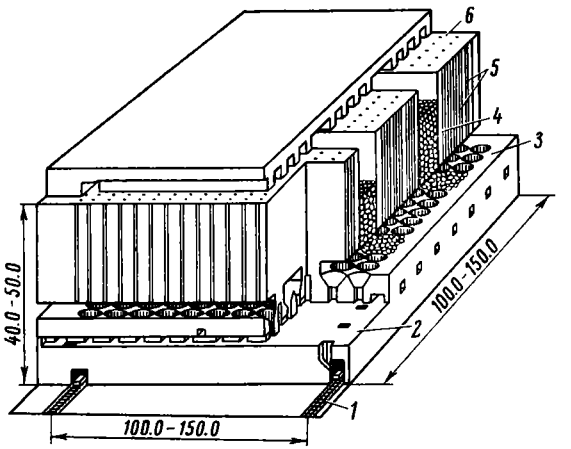


Fig. 245. Induced block caving by means of long vertical holes:
1—haulage drift; *2*—scram level: *3*—undercut level; *4*—compensation chamber; *5*—long hole; *6*—drill level

by drilling rounds of long holes therefrom. The width of the ore sections broken down into the compensation chambers is set at 18-20 m. The broken ore in a block is drawn off after the cover rocks have been caved. Induced block caving into vertical compensation chambers was successfully employed at the Sokolny mine (the Kazakh SSR). At first the ore was broken by means of shot-drilled down-holes. The base of a block was undercut to a height of 2 m by driftage of undercut openings or enlarging (by firing sectional-drill steel holes) the top portion of draw-holes to form funnels, directly under the solid ore, which was then pierced with long holes. Later on, shot drill rigs were supplanted by more efficient rotary-percussion and rolling-cutter drill units. In 1953-1960 a more perfect modification of the system, which involved breakage of ore by upward blast-holes, was devised and introduced at the mines of the Leninogorsk lead-and-zink combine (the Kazakh SSR).

This variant abruptly reduced (by 40-50%) the amount of subsidiary development work and made possible more effective utilization of drilling equipment.

A further step in the development of this system is the utilization of mobile equipment, which appreciably simplifies the preparation of a block and greatly enhances the efficiency of the method. The Institute for Projecting Non-Ferrous Metal Industry Enterprises

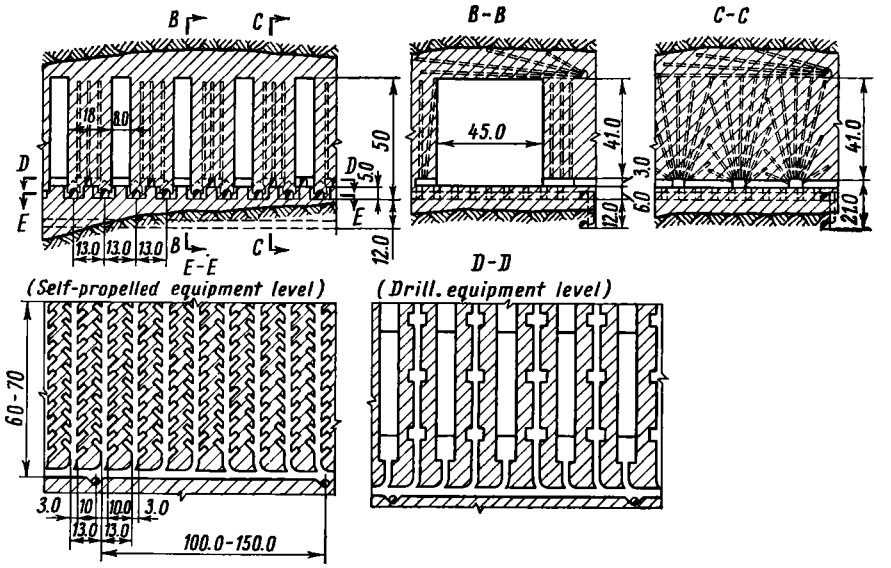


Fig. 246. Induced block caving into vertical compensation chambers with the use of mobile equipment

has elaborated a variant of induced sublevel caving with special reference to the working conditions at the mines of the Leninogorsk combine, taking account of the latest achievements and envisaging the use of mobile equipment on the drawing-off level (Figs. 246, 247). This variant can also be modified to provide for ore discharge on the haulage horizon (by combining the drawing-off and haulage levels). Mobile mine rigs may be used to drill long up-holes. These units are set up in drill workings driven 6 m above the floor of the mobile equipment level. Upward blast-holes can be charged with the aid of pneumatic charging devices which permit charging directly from the haulage horizon. Haulage roadways and crosscuts have a section of 9 sq m, drawing-off workings 4-5 sq m, and pull holes 4 sq m.

	Ordinary variant	New variant with mobile equipment
Inventory ore reserves in a block, thous. tons	1,100	420
Amount of development work per 1,000 tons of inventory reserves, cu m	8	4-5
Amount of subsidiary development work per 1,000 tons of inventory reserves, cu m	34	15
Monthly yield of a block, tons	20,000	50,000
Productive life of a block, years	up to 4	up to 1
Output per faceman per shift, tons	—	35-40

The efficiency of this variant may be greatly improved as compared with the usual variant without the use of mobile equipment. The table above gives comparative data with special reference to the conditions prevailing at the Leninogorsk mine (as estimated by the Institute for Projecting Non-Ferrous Metal Industry Enterprises).

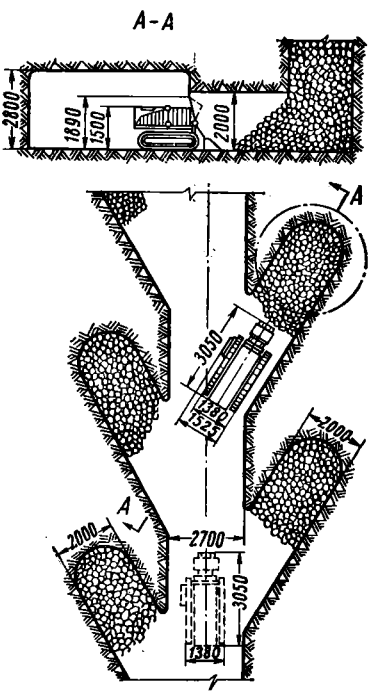


Fig. 247. Loading of ore from a drawhole with a loading-and-hauling machine

Induced Block Caving Without Compensation Chambers

It has been noted earlier that breakage of ore without compensation chambers (in tight faces) under appropriate conditions allows the amount of subsidiary development work to be reduced and the fragmentation of ore improved. An example is the apatite deposit at the Kirov mine (USSR), where ore is successfully mined level-wise by firing chamber charges without compensation chambers (Fig. 248).

The apatite occurrence at the Kirov mine represents a sheet-like lens of considerable size dipping at an angle of 25-30°. The hardness of ore f = 5-9, that of host rocks f = 10-12. The ores and enclosing country rocks are jointed, but sufficiently stable.

The ore in a level with an interval height of 40-60 m is taken out in blocks extending for 64 m on strike and 150-200 m across the strike. A block is prepared for stoping by driving a series of workings excavated on the haulage, breaking and undercutting levels. An undercut opening which measures 2 m in height is made by driving drifts and splittings (in undercutting, pillars are left whose area comprises 15-20% of the total undercut area). The draw-holes between the breaking and undercutting horizons are belled out to form funnels 6-6.5 m in diameter. The main development workings in the haulage level are a lateral drift (*1*) and haulage crosscuts (*2*) driven every 22-23 m. Above the undercutting horizon the block is divided

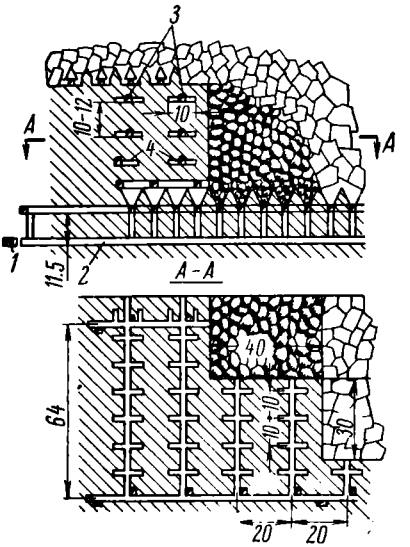


Fig. 248. Block caving induced by firing coyote charges

by subdrifts (*3*) into sublevels with an interval height of 7-12 m, and powder splittings (*4*) are excavated from subdrifts. Chamber charges are arranged on a 10 × 10 m pattern. The ore in the block is broken out in strips up to 40 m wide running on strike. The strip is extracted in vertical sections extending for one or two sublevel intervals, or throughout the entire level interval. Prior to breakage of ore the pillars abandoned in the undercutting level are blasted out. Adjacent strips are worked out with a lag of 30-40 m. Individual chamber explosive charges amount to 300-1,500 kg. The total weight of explosives per section reaches 20 tons, ammonite consumption comprising 0.35-0.5 kg per ton of ore. The ore is pulled in such a way as to provide for its plunging into the caved ground. The average yield of a block is 20-30 thousand tons of ore per month. Ore losses are 8-20%, dilution 7-8%. About 50% of the ore mined is undiluted. Output per miner in a producing section is up to 20 tons per shift.

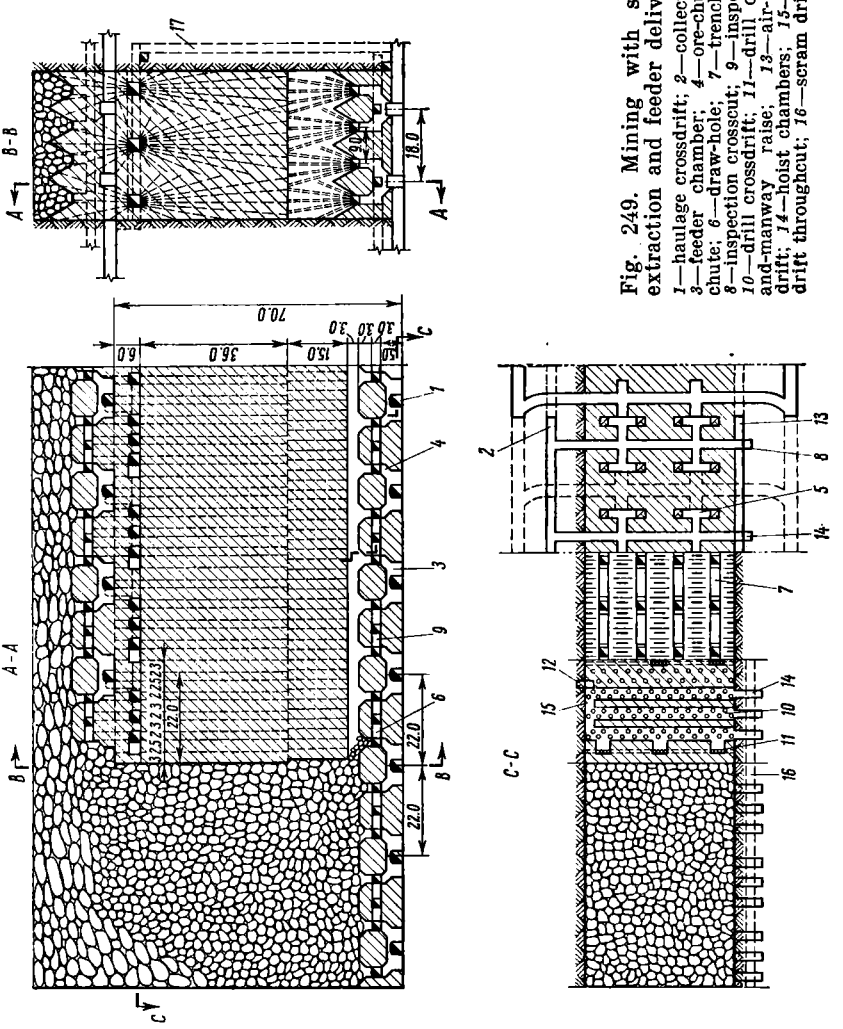


Fig. 249. Mining with single-stage extraction and feeder delivery of ore:
1—haulage crossdrift; *2*—collecting airway; *3*—feeder chamber; *4*—ore-chute; *5*—ore-chute; *6*—draw-hole; *7*—trench crossdrift; *8*—inspection crosscut; *9*—inspection drift; *10*—drill crossdrift; *11*—drill cut; *12*—air-and-manway raise; *13*—air-and-manway drift; *14*—hoist chambers; *15*—drill crosscut through; *16*—scram drift; *17*—raise

At the Kirov mine successful experiments were made on breaking out ore in a tight face between upward blast-holes (a section of ore to be broken down was 8.5 m thick, 26 m high and 32 m wide; blast-holes had a diameter of 150 and 165 mm, they were arranged fan-wise with a distance of 5 m between their ends).

Mining with tight-face breakage using long holes finds ever-increasing application at the Kirov mine.

Mining with single-stage extraction and feeder delivery of ore. This highly efficient system is very popular in Siberian iron ore mines in the exploitation of thick, steeply dipping deposits (Fig. 249).

The basic parameters of the method are:

Level interval	70 m
Floor height	14 m
Height of bench broken down by means of upward blast-holes	15 m
Ditto by means of downward blast-holes	36 m
Distance between haulage crossdrifts	22 m
Width of extraction block	up to 40 m, depending on block thickness
Distance between feeders	18 m

A distinguishing feature of the method is the absence of compensation chambers. The ore is broken down in a tight face (onto the caved ore). The required compensation volume is formed by the shifting and compaction of the tight surrounding material under the effect of a blast. The ore is broken in vertical slices 9-10 m thick pierced by several rows of blast-holes fired with millisecond delay. Continuous drawing of ore is assured by the use of reciprocating plate feeders with a rated capacity of up to 600 tons/hr (Fig. 250).

The basic technical and economic characteristics of the system are:

1. Length of development and subordinate workings per 1,000 tons of ore mined	2.3-2.5 m
2. Explosives consumption per ton of ore, g	600
3. Daily mining intensity, tons/m^2	2
4. Labour productivity per miner, tons/shift	
in drawing and loading of ore	250
in over-all operations	40-45
5. Ore losses	9%
6. Dilution of ore	18%

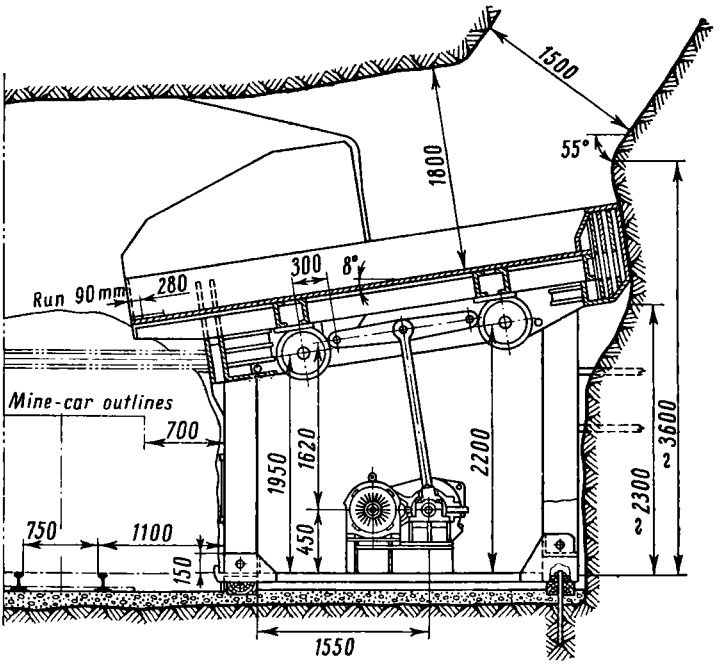


Fig. 250. Plate feeder setup in a block

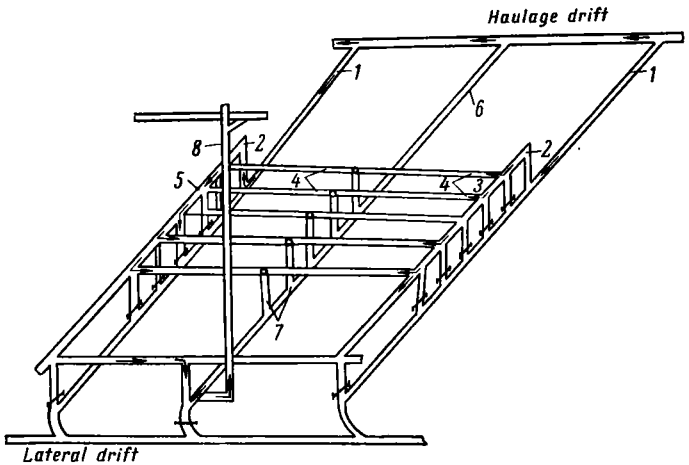


Fig. 251. Aeration of blocks in induced block caving:

1—haulage crossdrift; *2*—manway; *3*—ore-chute; *4*—slushing drift; *5*—connecting crosscut; *6*—ventilating crossdrift; *7*—air breakthrough; *8*—air raise

Ventilation in Block Caving Methods of Mining (General)

In block caving methods the ventilation system should provide good aeration of work places on the haulage, breaking or slushing and undercutting horizons.

Ventilation is accomplished by using stone workings (drifts and raises) and openings excavated in ore on the corresponding horizons. The mine openings are swept by main air current ("through" ventilation) with occasional use of booster fans.

Figure 251 represents a skeleton block ventilation scheme in block caving which is adjusted to fit the mining conditions and parameters of the methods actually employed. The air route is as follows: haulage crosscut, manway, connecting crossdrift, slushing drift, ventilation breakthrough, ventilating crossdrift, and raise. More detailed information on aeration of blocks in block caving may be found in the relevant courses of mine ventilation.

The following general conclusions can be made about the methods of induced block caving: they are highly efficient systems; as regards their characteristics they differ little from spontaneous block caving, but are more flexible as far as the conditions of their application are concerned.

Large-scale utilization of upward blast-holes allows the systems to be so designed as to minimize the amount of subsidiary development work and simplify the general preparation of blocks for stoping.

The major steps in raising the efficiency of these methods are:

1. Ensuring efficient primary haulage and subsequent transport of broken ore (potentialities of drilling and blasting operations in bulk breaking of ore by longholing are practically unlimited). The haulage efficiency can be substantially increased by employing heavy-duty scraper hoists, conveyers and self-propelled equipment; the use of the latter enables the ore to be loaded directly from the haulage horizon and thus greatly simplifies the preparation of blocks for stoping. High efficiency of drawing with the ore loaded directly into mine-cars can be achieved by employing vibrating feeders.

2. Reduction of rock pressure on the breaking (slushing) and haulage horizons (which can be accomplished in small blocks) and intensive mining of these horizons from the centre to the flanks of the deposit.

3. To achieve the highest possible recovery of metal from the mined ores all diluted ore (constituting around 20-30% of the mined bulk) should be passed through concentration plants. At non-ferrous metal mines flotation of mined ores should, conditions permitting, be supplemented by preliminary beneficiation of lean ores in heavy suspensions to separate a considerable proportion of gangue. A combination of ore extraction by bulk caving methods with efficient concentration of diluted ores is a measure of cardinal importance.

Major Advantages and Shortcomings of Mining Methods with Bulk Caving of Ore and Host Rocks

The main advantages of these methods are: high labour efficiency of facemen; low explosives and timber consumption; limited volume of development and subsidiary development work per 1 ton of ore extracted; low net cost of stoping; the possibility of achieving high intensity of mining.

Using the methods with block and sublevel spontaneous and induced caving of ore it is possible to attain the highest labour efficiency and the lowest cost of ore mined.

The chief drawbacks of these methods are: considerable losses and dilution of ore; subsidence and caving of the ground surface and enclosing country rocks (if it causes material damage); rigid requirements and, consequently, restricted application of the spontaneous block caving system. The technical and economic characteristics of the methods of mining with caving of ore and host rocks are listed in Table 24.

Table 24

Technical and Economic Characteristics of Systems with Caving of Ore and Host Rocks

Mining methods	Output per faceman per shift, tons	Coef. of recovery	Dilution factor	Timber consumption, m^3/ton	Explosives consumption, kg/ton
1. Sublevel caving with cushion . .	12-15	0.90-0.93	0.07-0.10	0.02-0.03	0.4-0.5
2. Sublevel caving without cushion . .	30-60	0.85-0.88	0.10-0.15	0.008-0.01	0.2-0.4
3. Spontaneous block caving of ore . . .	60-80	0.80-0.85	0.15-0.20	0.003-0.005	0.1-0.3
4. Induced block caving of ore by longholing	50-70	0.85-0.90	0.10-0.15	0.002-0.003	0.3-0.6

Note: 1. The coefficient of recovery and dilution factor do not take into account the useful components content in the host rocks. If this content is considerable losses owing to increased dilution may be cut down appreciably.

2. The average monthly yield of a block in block caving methods may be set at 40-60 thous. tons (withdrawing of ore). By using heavy-duty (up to 100-125 h. p.) scraper hoists or conveyers the yield of a block can be raised to 100-150 thous. tons per month (up to 2-3 tons of mineral extracted per 1 sq m of block per day).

3. Output per faceman in sublevel caving without a cushion amounts to 30-40 tons in variants with ore breakage by firing sectional-steel holes, about 50 tons in longholing, and 60 tons in spontaneous caving.

5-9. COMBINED METHODS OF MINING (CLASS VIII)

The combined methods are methods of mining thick deposits where a level is divided into rooms and intervening pillars of similar size worked out simultaneously or successively by using different systems. When working deposits exceeding 15-20 m in thickness rooms and pillars are so arranged that their major axis runs across the strike. The width of rooms and intervening pillars taken out across the strike varies within a wide range (depending upon the mining and geological conditions and design characteristics of the method), usually fluctuating from 8 to 15-20 m and from 6 to 10-12 m, respectively. The ceiling in rooms is taken to be from 0.2-0.3 to 0.5-0.6 of the room width, depending on the stability of the ore. The thickness of the room floor varies from 4-5 up to 10-15 m, depending on whether the ore is passed through a breaking or slushing horizon, or is discharged directly onto the haulage horizon. The ratio of ore reserves extracted from rooms and pillars in working across the strike commonly varies from 1 : 1 to 2 : 1.

Combined systems of mining should be characterized both in regard to extraction of rooms and recovery of pillars. Rooms are worked out by the following methods: shrinkage stoping, open stopes, cut-and-fill systems, and methods with timbering and filling of the stope area.

The methods employed in recovering intervening pillars in combined systems of room extraction are listed in Table 25.

The designations of combined systems of mining should reflect the methods of extraction of rooms and recovery of pillars (for example: "Shrinkage stoping of ore in rooms across the strike with subsequent backfilling of the rooms and topslicing of the pillars").

In practice combined systems carry names characterizing only the methods employed in working out rooms. Such names are incomplete and require additional explanation as to the recovery of pillars.

The design features of a method should take into account the optimal utilization of mine openings both in the extraction of rooms and in the robbing of pillars. To achieve the maximum effect in working out a room and a pillar, e.g. the block as a whole, rational widths of the rooms and pillars should be adopted.

In the foregoing we have set forth general information on combined systems of mining. Each of the methods included in a particular combination has been discussed in detail in one of the preceding classes of mining methods. The methods of mining in rooms across the strike described earlier in sections dealing with different classes are essentially methods of extracting ore in rooms by one combined

Table 25

Methods of Recovering Intervening Pillars in Combined Systems of Room Extraction

Mining methods used in extracting ore in rooms	Mining methods used in robbing intervening pillars	Notes
I. Shrinkage stoping of ore without subsequent back-filling of rooms	1. Block caving of pillars (prior to drawing of ore from rooms)	Drawing of broken ore from rooms and pillars is done simultaneously
	2. Topslicing of pillars from top down (when drawing ore from rooms)	—
II. Open-stope methods without subsequent back-filling of rooms	1. Block caving of pillars	Employed in working lean ores and to obtain filling materials
	2. Topslicing or sublevel caving after caving of room ceiling and adjacent ground	—
III. Shrinkage stoping or open-stope mining with subsequent backfilling of rooms	1. Timbering plus filling in short blocks or by vertical slicing	Used when caving of ground surface and host rocks is impossible
	2. Topslicing or sublevel caving	—
IV. Cut-and-fill methods	1. Topslicing	—
	2. Timbering plus filling in short blocks or by vertical slicing	Used when caving of ground surface and host rocks is impossible
V. Timbering-plus-filling methods (with ore extracted along the entire width and length of a room in slices from top down)	1. Timbering plus filling in short blocks or by vertical slicing	Used when caving of ground surface and host rocks is impossible
	2. Timbering-plus-filling methods of mining (with ore extracted along the entire width and length of a pillar in slices from top down)	

method or another. In Fig. 252 this is exemplified by combined shrinkage-stoping in rooms across the strike with recovery of pillars by sublevel caving simultaneously with drawing of ore from the rooms.

In choosing a combined system of mining consideration should be given to the fact that the stability of solid ore in intervening pillars usually decreases after the rooms have been worked out.

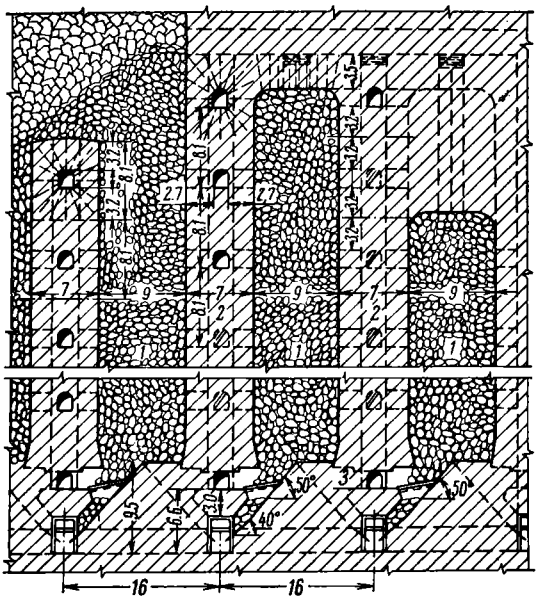


Fig. 252. Combined method of shrinkage stoping in rooms across the strike with pillars robbed simultaneously with the drawing of ore from the rooms (section along the strike):
1—room; *2*—intervening pillar; *3*—grizzly level

A combined system of mining should meet the same requirements as any of the methods discussed earlier (it should be efficient, safe, economical and productive).

In determining the technical and economic characteristics of combined methods of mining allowance should be made for the proportional share of the individual systems and their characteristics.

5-10. CHOICE OF A MINING METHOD

Selection of a System in Accordance with the Mining, Geological and Technological Factors; Technical and Economic Comparison of Possible Methods

The choice of the most effective mining method that would meet the basic requirements, viz. safety, low cost of operations, and efficiency, is one of the cardinal issues in the working of ore deposits. The matter may be approached in the following way. First, one should select the possible systems under the existing mining, geological and technical conditions. A special table could be used for the purpose; it should include the basic factors, their characteristics and the possible mining systems (Table 26).

In order to choose the appropriate method it is advisable first to list the classes of mining methods applicable under the given conditions and then proceed to selecting the possible methods within these classes.

The methods compatible with the particular factors may be conveniently designated in the table by conventional numbers assigned to each mining method. By studying these numbers it is possible to select methods adaptable to the specific conditions of a given deposit. The number of methods compatible with all the factors usually does not exceed two or three. In choosing a system one should take into account its design characteristics.

Due consideration for the design characteristics of a method (reduction in the room width, increase in the width or thickness of pillars, or abandonment of protective solid masses near exposed contacts of the ore body) makes it possible sometimes to select shrinkage or open-stope methods of mining with low-stability ores and host rocks.

The ultimate choice of the most effective method is made by considering the technical and economic characteristics. The selected system must meet all safety regulations.

A technical and economic comparison of the selected mining methods may be made by considering their economic efficiency, which should be assessed on the basis of the cost of the extracted useful components according to the state price rates and the operating expenses on the mining and processing of the ore. One should bear in mind that concentration and smelting costs often depend upon the efficiency of operations at the mine. Thus, failure to meet technological requirements as to the quality of the ore mined (high contamination with gangue, no separation into grades and classes) may sharply reduce the recovery of valuable components and raise the cost of processing and transportation.

Table 26

Choice of Mining Method

Item No.	Factor	Characteristics of the factor with reference to the working of the given deposit	Possible methods according to each factor
1	Shape of ore body		
2	Thickness of ore body		
3	Nature of contact between ore body and host rocks (and gangue inclusions)		
4	Dip angle of ore body		
5	Disturbances in occurrence of ore body—variations in dip angle, direction of strike, thickness; presence of faults and displacements		
6	Stability and other physical properties of ore (hardness, water-abundance)		
7	Stability and other physical properties of enclosing country rocks (hardness, water-abundance)		
8	Distribution of ore minerals in ore body (gangue inclusions, alternation of areas of rich and poor ore, presence of different grades of ore, as required by technological procedures)		
9	Mineralogical composition of host rocks (metallization of enclosing country rocks)		
10	Value of ore		
11	Susceptibility of ore to self-ignition, oxidation, and packing		
12	Depth of mining of ore body (increase in pressure, sudden outrushes of ore from pillars at great depths)		
13	Hydrogeological conditions of mining (water-bearing capacity of ore, host rocks, and of adjacent solid ground in place)		
14	Possible disturbances of ground surface and enclosing country rocks		
15	Miscellaneous factors (availability and state of communications, labour situation, electric power supply, distance to ore processing plants)		

A technical and economic comparison of mining methods proceeding exclusively from the efficiency of operations at the mine would be incomplete. It can only be adequate if the useful mineral won at the mine is the end product delivered directly to the consumer.

The economic efficiency of a mining system is estimated by using the basic formula

$$D = V - U$$

where D = economic efficiency in terms of roubles;
V = cost of useful component recovered from inventory reserves of solid ore in situ at state wholesale price, roubles;
U = all operating expenses charged against reserves of solid ore in situ, roubles.

Below we give simplified formulas for estimating comparative economic efficiency of commercial utilization of ore won by different mining methods (with permissible degree of accuracy):

$$V = \frac{a_s \varepsilon_m \cdot \varepsilon_p Q}{100^2 a_p} P$$

where a_s = mean content of useful component in solid ore in place, %;
ε_m = recovery of useful component in mining, %;
ε_p = recovery of useful component in processing, %;

$$\varepsilon_p = \frac{\varepsilon_c \varepsilon_s}{100}$$

ε_c = recovery of useful component during concentration, %;
ε_s = recovery of useful component in smelting, %;
a_p = content of useful component in finished product (for example, content of pure copper in blister copper), %;
Q = inventory reserves of solid ore in place, tons;
P = cost per unit finished product at state wholesale price, roubles;

$$U = U_1 + U_2$$

where U_1 = expenses on mining, ore beneficiation and transportation to processing plant charged against inventory reserves of solid ore in place with allowance for losses and dilution;
U_2 = expenses on smelting charged against inventory reserves of solid ore in place with allowance for losses;

$$U_1 = \frac{(U_m + U_c + U_t) \cdot Q \cdot C_r}{(1 - F_d)} \text{ roubles}$$

where U_m = cost of mining 1 ton of ore "free in surface bin", roubles;
U_c = cost of concentration of 1 ton of ore, roubles;
U_t = cost of transportation of 1 ton of ore from mine to concentration plant, roubles;
C_r = coefficient of ore recovery in mining (in fractions of a unit);
F_d = ore dilution factor in mining (in fractions of a unit);
Q = inventory reserves of solid ore in place, tons;

$$U_2 = \frac{a_s \cdot \varepsilon_m \cdot \varepsilon_c \cdot Q\,(U_s + U_{tc})}{100^2 a_{conc}} \text{ roubles}$$

where a_s = content of useful component in solid ore in place, %;
ε_m = recovery of useful component in mining, %;
ε_c = recovery of useful component during concentration of ore at plant, %;
a_{conc} = content of useful component in concentrate, %;
U_s = net cost of smelting charged against 1 ton of concentrate, roubles;
U_{tc} = net cost of transportation of 1 ton of concentrate from the concentration plant to smelters, roubles.

In making calculations individual figures may be adopted on the basis of the practical experience of certain enterprises, or taken tentatively. Tentative estimation is often admissible, for instance in evaluating deposits under exploration, since the initial data for comparison (reserves, ore quality, mining and geological characteristics) are also inaccurate. A more precise comparison of the possible mining methods can and must be made at a later date by projecting bodies on the basis of adjusted figures.

When no wholesale prices are available or when they are clearly incommensurate with actual expenditures, the systems may be compared according to the formula

$$C_2 - C_1 \geqslant P_{al} + P_{ad}$$

where C_2 and C_1 = first cost of end product in roubles for different mining systems;
P_{al} and P_{ad} = additional losses and dilution of ore for one of the methods compared.

The economic damage caused by losses and dilution of ore may be estimated by the following procedure proposed by M. Agoshkov, Corresponding Member of the Academy of Sciences of the USSR, with some simplifications introduced by the present author.

Estimation of Economic Damage Caused by Dilution of Ore

The economic damage resulting from excessive dilution when using one of the systems compared, in terms of 1 ton of commercial ore, can be determined by using the formula

$$D_{ex} = X_{ex} \cdot (C_m - C_{sys} + C_t + C_p)$$

where X_{ex} = excessive volume of gangue obtained with a system involving higher dilution per 1 ton of commercial ore

$$X_{ex} = \frac{P}{q_2} - \frac{P}{q_1}$$

where P = metal content in commercial (non-diluted) ore;
q_2 = metal content in mined commercial (diluted) ore extracted by one method, and q_1 by the other method;
C_m = total cost of mining 1 ton of commercial ore at mine "free in surface bin" (C_m = from 1.4 to $2.6C_{sys}$, on the average $C_m = 2C_{sys}$);
C_{sys} = production cost of 1 ton of commercial ore within the framework of a given system ("free in ore-chute on haulageway");
C_t = cost of transporting 1 ton of commercial ore from mine to concentration plant;
C_p = cost of processing 1 ton of commercial ore into end product (concentrate or metal).

Estimating Economic Damage Resulting from Losses of Commercial Ore

The economic damage caused by losses in mining consists of unproductive expenditures on prospecting and exploration, the increase in the depreciation rate for the capital invested in the construction and maintenance of a mine, unproductive outlays for the development and winning of the lost ore, and the decrease in the income of the enterprise in consequence of the losses of ore.

One should bear in mind that the decrease in the income of an enterprise as a result of the losses of ore is particularly high when the mined ore contains a sizable proportion of valuable constituents.

The economic damage caused by excessive losses of ore when using one of the mining methods compared (per 1 ton of the extracted payable ore reserves) will be:

$$L_{ex} = \alpha_l \left(\frac{C_1}{C_2} - 1 \right) \text{ roubles/ton}$$

where α_l = economic damage caused by the loss of 1 ton of commercial ore;

C_1 = coefficient of recovery of commercial ore when using one of the mining methods;
C_2 = ditto for the other method.

As mentioned earlier, α_l consists of individual kinds of damage, viz.

$$\alpha_l = (\alpha_d + \alpha_{dep} + \alpha_{db} + \alpha_c) \text{ roubles/ton of commercial ore}$$

where α_d = dilution losses;
α_{dep} = depreciation losses;
α_{db} = development and breakage losses;
α_c = commercial ore losses;

Determination of α_d

$$\alpha_d = 0.01p - \alpha_{pm}$$

where p = metal content in 1 ton of commercial ore, %;
α_{pm} = prospecting costs per 1 ton of metal, depending upon its wholesale price in roubles;
$\alpha_{pm} = (0.03\text{-}0.15)\,\alpha_m$, roubles
where α_m = state wholesale price of 1 ton of metal;
(0.03-0.15) = proportion of prospecting costs in state wholesale prices,

0.03-0.05 (3-5%)	for most iron ores,
0.05-0.10 (5-10%)	for chromium, manganese and titanium ores,
0.08-0.12 (8-12%)	for ores of non-ferrous and precious metals,
0.12-0.15 (12-15%)	for ores or rare metals and trace elements.

Determination of α_{dep} (depreciation rate of capital outlays per 1 ton of commercial ore).

The proportion of α_{dep} in the total mining cost of 1 ton of ore is on the average 15%, while the accountable part of depreciation charged against 1 ton of commercial ore reserves averages 5% of the total prime cost. Then

$$\alpha_{dep} = 0.05C_m \text{ roubles/ton of commercial ore reserves.}$$

Determination of α_{db} (the cost of development work and of actual breakage per 1 ton of commercial ore).

This may be set on the average at $\cong 40\%$ of the total mining costs, then

$$\alpha_{db} = 0.40C_m \text{ roubles/ton of commercial ore reserves}$$

where C_m is the cost of mining.

The effect of losses on the cost of development and stoping is fully reflected in the cost of 1 ton of commercial ore mined, and through it in the cost of 1 ton of extracted commercial ore reserves. Therefore α_{db} should not be taken into account in determining α_l.

Determination of α_{com} (economic damage caused by the reduction in the profits per 1 ton of the lost commercial ore).

This can be estimated by using the formula

$$\alpha_{com} = 0.01o \cdot i \cdot \alpha_{met} - (C_m + C_{pr}) \text{ roubles/ton of commercial ore}$$

where o = metal content in the lost ore, %;
i = coefficient of metal recovery in processing;
α_{met} = state wholesale price per 1 ton of metal, roubles;
C_m = total cost of mining 1 ton of commercial ore, roubles;
C_{pr} = cost of processing 1 ton of commercial ore, roubles;
"i" varies from $0.85 \times 0.85 = 0.72$ to $0.95 \times 0.95 \cong 0.9$.

In practice it is often difficult to obtain data on smelting and in such cases economic damage should be estimated with special reference to the production of concentrate; then α_{met} will be the state wholesale price of the concentrate, "i" the coefficient of metal recovery in beneficiation (from 0.85 to 0.95), C_{pr} the cost of processing including transportation. In order to correlate the reduction in profits with the current period it is necessary to insert the time factor in the denominator

$$C_{com} = (1 + \varepsilon)^t$$

where ε is the efficiency factor for a given industrial branch (for the mining industry it may be set at 0.1). With $\varepsilon = 0.1$, $C_{com} = 1.1^t$, where t is the period of mine exploitation with delivery of ore having an average metal content "p".

With C_{com} taken into account the reduction in the profits per 1 ton of the lost commercial ore will amount to

$$\alpha_{com} = \frac{1}{1.1^t}[0.01p \cdot i \cdot \alpha_m - (C_m + C_{com})] \text{ roubles/ton}$$

When the available ore reserves ensure exploitation of a mine for a long period of time (25-30 years) the economic damage caused by the reduction in the profits decreases considerably (the value of the term $\frac{1}{1.1^t}$ greatly diminishes).

Calculations prove that economic damage caused by the reduction in the profits may be disregarded for ores with a low metal content and for long periods of mine exploitation (over 20-25 years). The above formulas for estimating the economic damage caused by the losses and dilution of ore are somewhat inaccurate, since they underrate metal recovery from the ore in processing mined and lean ores, etc. However, attempts at more precise calculations will not be justified because of the limited accuracy of the initial data.

Below we give some information which can be used in tentative evaluation of deposits under prospecting.

Estimation of the Total Cost of Mined Ore

Tentative determination. The total prime cost of mined ore can be tentatively estimated on the basis of available data as to the total prime cost obtained by another mining method, proceeding from the following percentage relationships for various systems of mining or from the prime cost of stoping (see below).

Comparative table of total prime cost of mining 1 ton of ore	% %
Mining methods with timbering and filling	100
Cut-and-fill methods	80-85
Shrinkage stoping with ore breakage by blasting short holes	60-70
Shrinkage stoping with bulk breakage of ore	25-30
Open stope methods	
thin ore bodies	70-80
thick ore bodies	30-50
Topslicing method	65-75
Sublevel caving with mat	50-60
Sublevel caving without cushion and with mass breakage of ore	25-30
Spontaneous block caving of ore	15-20
Ditto with induced caving of ore	20-25
Methods of mining very thin ore bodies with selective extraction of vein matter and host rocks	150-250

The comparative prime cost of winning ore by combined methods of mining should take into account the proportional share of the component systems.

The determination of the prime cost of mining ore on the basis of cost figures for another system in accordance with the indicated percentages is possible on condition that the production capacity of the mine, water inflow, depth of mining, state of mechanization and other operating conditions not depending upon the mining method are approximately the same in both systems. If the mining conditions are different, the necessary corrections should be introduced.

Determination of the prime cost of mining 1 ton of ore according to the proportional share of stoping costs. The mining costs per 1 ton of ore

$$C = r + d \text{ roubles}$$

where r = running expenses per 1 ton of ore, roubles;

d = depreciation of invested funds per 1 ton of ore, roubles.

The proportional shares of the various expenditure items in the estimate of the prime cost of ore may be distributed as follows: running expenses 80-90% of the total cost of ore mining; depreciation of invested funds 10-20% (the higher percentage should be adopted

for a mine with a limited service life, low production capacity and a substantial volume of major permanent workings per 1 ton of ore mined).

Running expenses, in their turn, consist of the following items:

	%%
1. Stoping	40-60
2. Development work	8-15
3. Haulage	5-10
4. Hoisting	4-8
5. Pumping	0-4[1]
6. Ventilation	1-2
7. Repairs and maintenance of mine openings	1-2[2]
8. Underground lighting	0.5-1
9. Ground-surface services	3-5
10. General shaft and mine expenses	15-25

From the above figures it follows that expenditures on stoping operations have the largest proportional share comprising 40-60% of all running expenses, or approximately 30-50% of the total prime cost of mining per 1 ton of ore.

With an increase in the output of a projected mine expenses per 1 ton of ore under items 1 and 2 vary insignificantly.

Expenses under items 3, 4, 6, 7, 8, and 9 decrease, but not in direct proportion, while expenses under items 5 and 10 reduce almost proportionally to the increase in the production capacity of the mine. On the average, a 10% increase in the annual production of small mines (with an output of less than 100 thous. tons per year) reduces the running expenses per 1 ton by 2.5-3%, and for large mines (with an output of over 500 thous. tons per year) by 1-1.5%.

The initial investments for the construction of mining enterprises, in their turn, consist of the following items:

Distribution of initial investments in %% of the total outlays for industrial construction	
Preliminary work	2-5
Major excavations of permanent mine-openings (sinking of shafts, driving of adits, crosscuts and other permanent workings)	35-60[3]
Electrical and mechanical equipment	20-40
Technical, administrative, auxiliary and transport structures and facilities	15-35
Minor property and stock	1-3

[1] In exceptional cases with a maximal water inflow — 9-10%.

[2] In especially difficult conditions of mining and in very unstable ground—up to 5-10%.

[3] On the average 40%.

Determining the cost of stoping. The cost of stoping includes: direct wages of facemen plus bonus and extra payments, the cost of electric power and compressed air, filling material, timber, explosives and other supplies necessary for stoping.

The largest item in the cost of stoping is the total wages of the faceworkers (40-60%).

A tentative estimate of the cost of stoping can be made by taking as a basis the output per faceman per shift, the total wages per faceman per shift (direct wages, bonus and all extra payments) and the proportional share of the total wages in the cost of stoping.

Determining the total production cost of ore. The total production cost of ore can be estimated from the proportional share of expenses on ore stoping in the total cost.

It has been noted earlier that this estimate is tentative, but this degree of accuracy is admissible, for example, in the preliminary evaluation of deposits under prospecting.

Approximate Determination of Concentration Cost Per 1 Ton of Ore

Concentration cost varies within a wide range and depends upon the ore composition, adopted beneficiation procedures, production capacity of the plant and the cost of reagents.

In tentative estimates the cost of concentrating 1 ton of monometallic ores by simple methods (flotation, gravitation, magnetic concentration, amalgamation) may be set at 0.4-0.6 of the full mining cost of 1 ton of salable ore. With complicated and expensive methods of beneficiation (combined procedures with the use of gravitation, electromagnetic and flotation methods) the cost of concentrating 1 ton of ore may reach 1.0-1.2 times the cost of mining 1 ton of salable ore. The smaller coefficient (0.4 or 1.0) should be adopted for a plant with a high production capacity (over 500 tons of processed ore per day). If the ore is sorted prior to its processing at the plant, its processing cost should include the expenses on such preliminary sorting. In this case, however, allowance should be made for the reduction in the amount of ore processed at the concentration plant. The cost of smelting of concentrates fluctuates within very wide limits and can be determined only from data supplied by individual plants. If no information on smelting is available mining methods should be compared merely on the basis of figures relating to the extraction and concentration of the ore.

The Use of the Profit and Loss Index

This index may be used not only in selecting the method of mining, but also to substantiate the acceptance of the bulk or the selective method for extracting a mineral, or the expediency of underground

or ground-surface sorting (the items to be determined are the cost of the recoverable useful constituents at state wholesale prices and all operating expenses, both of which in the final analysis indicate whether the given enterprise is a paying concern).

In justifying the choice of the bulk or the selective method of extraction it is necessary to compare the corresponding systems of mining with consideration given to additional expenses involved in selective extraction, to the various factors relating to the recovery of the useful constituents at a mill or plant, and to the different expenditures associated with bulk and selective mining.

The additional expenses in selective mining should include expenses caused by an increase in the amount of work in driving mine openings, their upkeep and sorting at the production face (if this is envisaged).

Efficiency of Sorting and Measures to Increase It

Economic efficiency of sorting can be determined tentatively by using the formula

$$D_{sort} = (U_1 + V_1) - (U_2 + V_2)$$

where U_1 = reduction in expenses on sorting (the cost of transportation and concentration of barren rock is eliminated), roubles;

V_1 = value of additionally recovered metal in sorting (increased recovery of the useful constituent from the sorted-out ore at a dressing mill or concentration plant), roubles;

U_2 = expenses on sorting, roubles;

V_2 = value of the metal lost in sorting (metal lost in discarded rock).

If the value of D_{sort} is negative sorting is inexpedient.

In solving problems of sorting the following approximate figures may be used (more exact figures are established in each individual case on the basis of special observations and investigations).

Underground sorting: minimal size of lumps in the muck to be picked out 75-100 mm; labour productivity per picker per shift from 1 to 2.5 tons of picked-out muck, depending upon the conditions of sorting; the amount of sorted-out barren rock from 20 to 50% (on the average 30-35%) of the total amount of waste mixed with ore.

Sorting on the ground-surface: minimal size of lumps in the muck to be sorted out 25-50 mm (usually 50 mm); labour productivity per picker per shift from 1.0-1.5 to 4-6 tons (on the average 2.0-4.0 tons/shift) of sorted-out muck, depending upon the conditions

of sorting; the amount of barren rock to be sorted out from 20-30 to 70-80% (on the average 50-60%) of its total amount in the muck; the mean content of useful constituents in the discarded rock is determined on the basis of special observations; it may tentatively be set at 5-10% of the content of useful constituents in the total muck to be picked; the major additional expenses on the organization of ground-surface sorting consist of pickers' wages, expenses on conveyer transport of the material undergoing sorting; reduction in the expenses as a result of ground-surface sorting (the cost of transporting discarded barren rock from the mine to the plant and of rock processing at the plant is eliminated).

The yield of lumps of different sizes can be determined from the sizing curve.

The basic measures to increase the effectiveness of ground-surface sorting are: preliminary crushing of ore down to 100-150 mm, washing of ore before sorting; conveyer belts 600-700 mm wide when the pickers work on one side of the belt and 1,000-1,200 mm when they work on both sides; conveyer belt speed 0.2-0.3 m/sec; distance between pickers 0.9-1.8 m; intensified and special lighting of the muck to be sorted; collecting ore wash water in settling tanks with subsequent utilization of the slime for the recovery of metal; periodic, well-organized observations of pilot sorting, which is to furnish all the necessary information.

In selecting the type of sorting preference should generally be given to ground-surface sorting as the more effective method. But even then additional underground sorting is also possible.

Extensive research into the problems of sorting has been carried out at the Skochinsky Institute of Mining. The results of these investigations are published in a book by A. Nazarchik *Razubozhivaniye rudy pri razrabotke zhilnykh mestorozhdeniy* (Dilution of Ore in the Working of Lode Deposits), Akademizdat, 1960. According to the book the cost of ground-surface sorting per 1 ton of discarded barren rock largely depends on the efficiency of pickers. This, expressed in tons of picked-out muck (rock), is contingent upon the size of lumps and comprises 0.5 ton/shift with lumps measuring 25 mm, 1 ton/shift for 50 mm, 2 tons/shift for 100 mm, 3.5 tons/shift for 150 mm, and 7 tons/shift for 200 mm.

5-11. MEASURES TO PROMOTE THE EFFICIENCY OF MINING ORE DEPOSITS

1. Extensive introduction of measures based on the experience of Soviet collieries. Change-over, wherever possible, to extraction of ore with longwalls, the use of metal posts and metal netting, movable shield support, electric auger drilling in grounds of limited

strength, broad utilization of conveyers, high advance rate in driving major mine openings—shafts, crosscuts, adits, and introduction of cyclic operation. Particular attention should be given to cyclic work schedules in stoping.

2. Wide utilization of multiface, multimachine drilling and of other methods applied by progressive workers.

3. All-round mechanization of operations, from work at the production face to storage in bins on the ground-surface, with extensive use of remote and automatic control of individual machines.

4. Broad application (under suitable conditions) of mining methods with mass breakage of ore by longholing. This is one of the promising methods meeting technical, economic and hygienic requirements in stoping. Tight-face breakage of ore should be practised where the conditions allow it.

5. Wide use (with all mining systems) of self-propelled drilling, loading, and haulage equipment, especially in working flat and gently sloping deposits. Utilization of self-propelled equipment along with deep blast-holes is of paramount importance in mining ore deposits.

6. Increasing the effectiveness of cut-and-fill methods of mining through the use of float-fill and stowing machines, self-cementing packing and quality floorings.

7. Mechanization of concrete mixing and placement at mines with broad utilization of self-propelled units for mixing and feeding concrete.

8. Large-scale introduction of rock bolting to provide support in development and productive workings.

9. Introduction of flexible roofing in topslicing.

10. Accurate recording of losses and dilution and carrying out all possible measures to reduce them.

11. Determination of the efficiency of mining methods by considering the mining costs, losses, dilution of ore, and figures characterizing the subsequent technology of ore treatment at concentration plants. It should be borne in mind that failure to meet technological requirements (diluted ore not graded and sorted, etc.) often leads to a decrease in the recovery of useful components and an increase in the cost of ore treatment.

12. Establishing more accurate mining and geological characteristics of deposits to be worked. Insufficient degree of exploration of deposits (including characteristics of host rocks) is one of the causes responsible for errors in projects and the resulting inefficient exploitation of deposits.

13. Selection of appropriate design features for combined systems of mining and striving for maximum efficiency in the working of the entire block (efficient extraction of rooms and pillars).

14. Radical improvement in stoping technology with provisions made for a considerable rise in the intensity of stoping.

15. Large-scale investigations into mass breakage of ore, setting up rational parameters for mining methods, establishing expedient methods of rock pressure control, reducing losses and dilution, dust control in stoping, introducing effective measures to combat fires in chalcopyrite mines, carrying out research in connection with all-round mechanization and automation of mining operations, devising and applying new types of support and methods of ore breakage.

16. Introducing, under suitable conditions, a basically novel method of mining—underground leaching of solid ore in place without preliminary crushing (sandy ores), or with preliminary breaking of solid ore in place (longholing with the use of ordinary or, in the future, nuclear explosives). Underground leaching has been introduced in copper and uranium mines.

PART TWO

THE OPENCAST WORKING OF MINERAL DEPOSITS

I. QUARRYING OF ORE AND SHEET DEPOSITS

CHAPTER VI

GENERAL INFORMATION, BASIC DEFINITIONS AND TERMS

6-1. GENERAL

Quarrying of mineral deposits dates back to ancient times, but before the XX century the scope of opencast mining had been insignificant and it was not until the XX century that immense mining enterprises began to appear, some of them yielding tens of millions of tons annually.

At present about 50% of all useful minerals produced in the USSR are obtained by opencast mining. In the future opencast mining will account for 80% of the total production of salable iron ore, up to 65% of the aggregate tonnage of non-ferrous metal ores, and more than 50% of the coal output (Table 27).

Table 27

Proportional Share of Opencast Mining in the Total Production of Minerals, %%

Mineral	Years				
	1950	1960	1965	1970	1980
Coal	11	20	24	28	52
Iron ore	44	56	63	70	80
Ores of other metals	46	53	61	64	70
Manganese ore	—	29	55	75	80
Mined chemical raw materials	—	50	72	78	80
Building materials	100	100	100	100	100

The major points characterizing the high technological and organizational level of opencast work are: all-round mechanization of mining operations with the use of powerful and highly efficient

equipment, such as excavators, drill rigs, electrical locomotives, dump-trucks, conveyers, hydraulic giants, etc.; application of effective mining methods; working on schedule; high performance of a number of enterprises attained in short periods of time.

Advantages of opencast work. The great popularity of opencast mining is due to its substantial advantages over underground work, greater possibilities of mechanization, the use of large powerful and highly efficient machines with any type of drive; greater safety and freedom of movement, sufficient light and air and, consequently, improved hygienic conditions and higher efficiency of labour; the possibility of quicker increase in the production of minerals, the higher rate of their recovery, the greater productivity of labour at a lower cost. At many Soviet open-pits the output per man is five to six times as high as in underground mines, while the mining cost per ton of the mineral is 25-30 per cent of that in underground mines; initial investments per unit capacity are also much smaller. Calculations made by projecting organizations showed that initial investments per unit capacity in the construction of new, highly efficient open-pits in the Kemerovo and Krasnoyarsk areas will be approximately 10 per cent of those for underground mines.

Drawbacks of opencast work. Opencasting is not devoid of certain shortcomings, such as considerable original capital outlays for the purchase of equipment and excavation of spoil (stripping operations), dependence upon climatic conditions, etc.

In spite of these disadvantages opencast work, when done under favourable conditions, is more effective than underground mining.

6-2. BASIC CONCEPTS AND MINING TERMINOLOGY

Open-pit, open-cut or **quarry**—a mining enterprise engaged in the surface working of a mineral deposit. The name is also given to an aggregate of open mine workings used for the extraction of a mineral.

Thus, the term has two meanings—economic and technical.

Open-pit field (take)—a deposit (or part of it) worked by one open-pit. It is divided into horizontal layers—horizons, which assume a stepped (benched) shape in the course of mining.

Benching bank, or simply **bank,** or **bench**—a layer of barren rock or useful mineral worked by independent means of breaking, loading, and transportation. Each benching bank is assigned an absolute or relative elevation mark for its bottom, or bench floor, or toe (elevation mark for transportation routes). The benching bank consists of the elements shown in Fig. 253.

Top (roof)—the upper (usually horizontal) part of the bench surface.

Bottom (bench floor, or **toe)**—the lower (usually horizontal) part of the bench surface.

High wall, or **slope**—a front upright or, more often, sloping surface bounding the bench on the side of the worked-out space.

Angle of slope, or **angle of high-wall**—the angle formed by the bench slope with the horizontal plane.

Bench production face—surface undergoing mining (the side face or high-wall of a bench).

Open-pit sides—side surfaces bounding an open-pit (Fig. 254). Hence, the open-pit sides represent an aggregate of high-walls

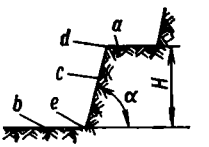


Fig. 253. Elements of an open-pit benching bank:
a—top; *b*—bottom (bench floor, or toe); *c*—high wall; *d*—bench crest, or edge; *e*—toe of the high wall; α—angle of slope; *H*—bench height

(slopes) and ledges on all benching banks of the open-pit. One should distinguish the open-pit side in production, where mining is underway or has not been completed, and the virgin (non-productive) side, where no mining operations are conducted (or they have been completed).

Berms—horizontal ledges situated on the non-productive side of the open-pit and serving to accommodate transportation roads

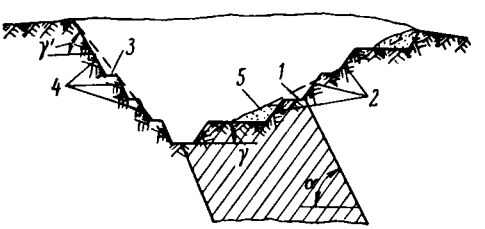


Fig. 254. Open-pit layout (cross section):
α—dip angle of bed; γ—angle of slope of the open-pit side; γ'—angle of slope of non-productive (virgin) side of open-pit; *1*—working bench; *2*—open-pit side in production; *3*—protective berm; *4*—virgin, or non-productive side of open-pit; *5*—mined bulk

(transport berms) or to protect the operating equipment from falling lumps of ground (protective berms).

Working area, or **site**—a horizontal ledge on the production side of an open-pit serving to accommodate drilling, mining and transport equipment.

Open-pit floor—horizontal surface of the quarry bottom.

Bench crest, or **edge**—the line of intersection of the quarry side with the subjacent unmined surface. This line defines the upper contour of the open-pit.

Toe of the high-wall—the line of intersection of the quarry side with its floor. This line defines the lower contour of the open-pit.

Open-pit side slope—a plane tangential to the crest and toe of the open-pit.

Slope angle of the open-pit side—an angle formed by the open-pit side slope line with its projection on the horizontal plane.

Trench—an open excavation of trapezoidal cross section. A trench carried along naturally sloping ground has an incomplete cross-section. Such an excavation is called a **semi-trench.**

Three types of trenches are usually used in a quarry.

Ingoing, entrance, or **opening ramp trench,** sometimes also called a **through,** or **box cut**—a trench securing access from the ground

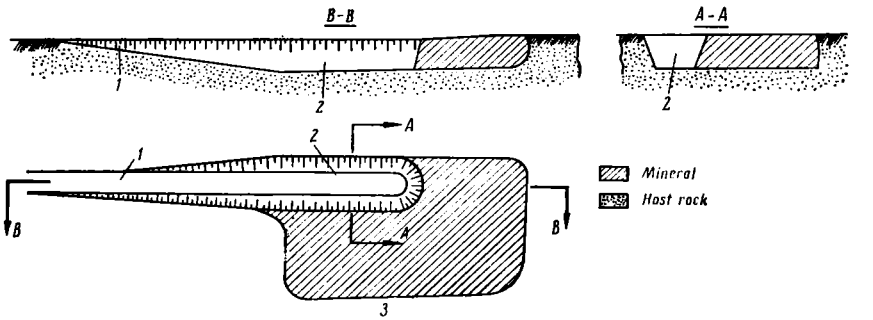


Fig. 255. Layout of trenches in a flat deposit of limited thickness with one-way mining:

1—outgoing, or exit ramp trench; *2*—working trench; *3*—deposit contours

surface to working horizons of an open-pit, or approach from one of the active parts of a deposit to another. Ingoing ramp trenches have a down- or up-grade and their width should conform to the type of transport facilities employed (Fig. 255).

Working trench—a trench ensuring initial front for working a bench. Working trenches are started from ingoing ones. They run horizontally and, less frequently, with an easy slope. These trenches cut a deposit in a pre-set direction, usually from one of its boundaries to the other (Fig. 255).

After a working trench has been made one or both of its sides are worked (shaping of the bench).

Cuts—parallel strips by which a bench is worked (Fig. 256).

Block—a part of a cut along its length worked out by independent means of breaking and loading. A long cut is divided into individual blocks in order to speed up the extraction of the cut (Fig. 256).

Working front, or face room of a bench—part of a bench along its length which is prepared for mining. Preparation of the working front consists mainly in driving a working trench, bringing in transport facilities and securing supply of energy which will ensure the

operation of the main equipment in blocks (locomotives, drill rigs, excavators). The working front of a bench is measured in linear metres. A normal working front per one excavator with railway, motor-truck and belt-conveyer transport is 100-200 m.

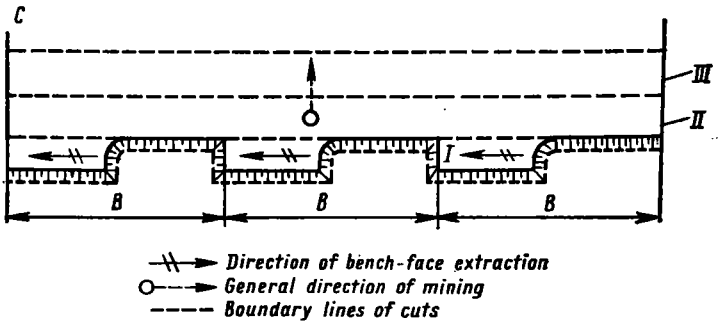


Fig. 256. Layout of cuts and blocks in extracting a strip (plan view):
I, II, III—cuts; *B*—blocks; *C*—open-pit contour

Working front, or face room of an open-pit is the total extent of the working fronts of the quarry benches. It is necessary to distinguish between a working front in ore and in ground.

6-3. DESIGN DIMENSIONS OF A QUARRY AND OF ITS INDIVIDUAL ELEMENTS

Dimensions of an open-pit field. These are often determined by the dimensions of the deposit itself. In planning an open-pit the optimal dimensions of the open-pit field are set so that the total sum of capital outlays per 1 ton of the mineral mined should be minimal, account being taken of the period over which initial investments are made. In the majority of Soviet open-pits the length of the open-pit field on strike is 2-2.5 km, rarely 4-5 km.

The extent of the field across the strike is, as a rule, smaller than its length and approaches it only in open-pit fields with a circular outline in the plan view. The dimensions of an open-pit field depend upon many factors, such as: the properties of the ground, the thickness and dip angle of the deposit, the mining depth, the local topography, the annual tonnage of the mineral quarried out and the amount of stripped spoil, the method of mining and opening up of the open-pit, the equipment employed, etc. V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR, cites the following data on the dimensions of open-pit fields in relation to the service life of the open-pit (Table 28).

Table 28

Types of Open-Pit Fields

Size of open-pit field	Type of opencast work	Open-pit area in plan, sq km	Ultimate depth of open-pit, m	Aggregate volume of mined bulk, mill. cu m	Service life of open-pit, years
1. Very small	surface	up to 0.4	up to 20	up to 10	up to 10
	upland	up to 0.3	up to 40	ditto	ditto
2. Small	surface	0.4-2.0	up to 40	10-100	10-25
	upland and below ground surface	0.3-1.5	40-100	ditto	ditto
3. Medium	surface	2.5-6.0	up to 60	100-500	15-30
	upland and below ground surface	1.5-5.0	100-200	ditto	ditto
4. Large	surface	4-20	up to 80	500-2,000	30-60
	upland and below ground surface	4-12	100-250	ditto	ditto
5. Very large	surface	10-40	up to 120	2,000-10,000	60-100
	below ground surface	10-30	200-800	ditto	ditto

Dimensions of a Bench and the Number of Benches in an Open-Pit

The adopted dimensions of a bench (its height and angle of slope) depend upon the equipment used, the nature of the ground to be mined, and the conditions of work.

The setting up of the correct relationship between the height of the bench and the working dimensions of the excavator ensures the most efficient, safe and economical operation of the equipment on the bench and the delivery of high-quality raw materials. The height of the bench is closely related to the digging height of the power shovel. In non-solid rocks it is commonly taken to equal the digging height. In ledge rocks the parameters of a bench are chosen so that the height of the broken rock pile scattered by blasting does not exceed 1.5 times the digging height of the power shovel.

Selective loading of different grades of ore or loading of ore separately from waste is possible only when the height of the bench does not exceed the digging height of the excavator.

In the practice of opencast work the bench height usually varies from 3 to 20 m (with power shovel loading usually 10-15 m for waste and 8-12 m for ore). Projects of new big open-pits envisage bench

heights of 12 to 15 m. The angles of working bench high-walls are set at 70-80° for igneous rocks, 50-60° for sedimentary rocks, 40-50° for semi-ledge and dry sandy grounds, and 35-45° for argillaceous rocks.

High-walls of worked-out benches (those of the non-productive side of the open-pit) have a lesser (by 5-10°) slope, since they continue to serve for a long time. Often these benches accommodate permanent open-pit roads.

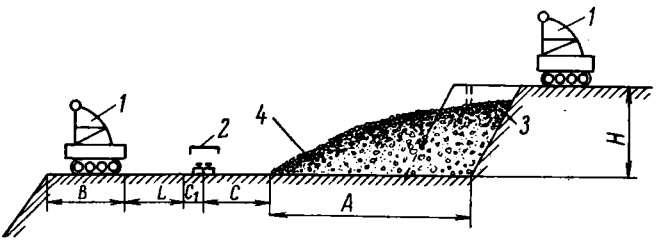


Fig. 257. Layout of a bench working area in a quarry:
1—drill rig; *2*—railway car; *3*—high-wall; *4*—pile of broken ore

The high-wall angle and the bench height depend on the water-bearing capacity of the ground. Dry rocks ordinarily permit of a greater slope angle and a higher bench.

The width of working bench areas varies within wide limits depending upon the equipment used and the mode of operation; it usually equals 40-60 m, but reaches 80-100 m when multirow borehole blasting is used.

The number of benches in an open-pit corresponds to the planned depth of its mining. In opencast work this depth may be very considerable at times.

The Korkino coal open-pits are among the deepest in the USSR (300 m). The new project for working this occurrence envisages its deepening down to 500 m below the ground surface. The ultimate depth of a series of iron ore open-pits is to reach 730 m: 725 m for the Kacharsk open-pit, 700 m for Sarbaisk, 560 m for Korshunovsk, etc. Among the foreign open-pits the copper pit "Utah-Copper" (USA) has the greatest depth (over 700 m).

The minimal width of a quarry bench working area (Fig. 257) in mining hard-ledge rocks can be estimated by the formula:

$$V_r = A + C + C_1 + L + B$$

where A = width of broken rock pile after blasting, m;
C = distance along bench toe from edge of broken ore pile to railway track axis (3-4 m);

C_1 = half-width of railway bed — 2.0 m with the 1,000 mm track gauge, 2.5-3.0 m with the 1,524 mm (standard) gauge;
L = marginal area to ensure extraction of prepared reserves in underlying bench (10-20 m);
B = width of strip of ground to be left for drilling operations (6-10-12 m).

The width of the broken rock pile after blasting depends upon the method of drilling and blasting. In the general case the width of the broken rock pile equals

$$A = C \cdot H + b\,(n - 1),$$

where H = height of bench, m;
C = coefficient (set at 1.5-2.0);
b = distance between borehole rows in multirow blasting, m;
n = number of fired borehole rows.

In the final stage of working quarry benches protective berms should be left with a width equal 0.1-0.2 of the height of the adjacent bench but they should not be less than 1 m. In deeper open-pits protective berms at least 6 m wide are left every 30 m.

Open-Pit Side Slope Angles

The slope angle on the non-productive side of an open-pit (reclamation angle) is determined with a view to ensure the stability of quarry sides, the formation of berms, and the accommodation of opening trenches.

The slope angle on the productive side depends upon the width of the bench working areas and the bench height.

On the basis of experience in open work by advanced methods and of an analysis of data used in planning opencast mining with the use of transport vehicles, which accounts for about 70% of the quarried tonnages, the following parameters are suggested (Table 29).

Commonly adopted slope angles for both productive and non-productive (worked-out) quarry sides are given below:

	deg
Productive open-pit side:	
Hard ledge rocks	18-20
Rocks other than hard ledge in ordinary conditions	15-18
Arrenaceous and clay rocks difficult to dewater	10-15
Non-productive open-pit side:	
Hard ledge rocks	40-45
Rocks other than hard ledge in ordinary conditions	35-40
Arrenaceous and clay rocks, difficult to dewater	20-30

Table 29

Parameters of Open-Pit Mining Methods (after N. Melnikov)

Characteristics	ЭКГ-4 power shovel:		ЭКГ-8 power shovel	
	without loosening of ground	with loosening of ground	without loosening of ground	with loosening of ground
1	2	3	4	5
Bench height, m	9.5-10.5	15.0-16.0	12.5-13.5	19.5-21.0
Width of cut, m	15.5-16.0	15.0-16.0	19.0-19.5	19.0-19.5
Width of broken rock pile scatter, m	—	30.0-32.0	—	38.0-39.0
Width of quarry bench working area, m . . .	60.0-62.0	75.0-78.0	68.0	87.0-88.0

Correct determination of slope angles on quarry sides is of great practical importance, for a 1° deviation in a slope angle changes the volume of the quarried ground by about 4%.

The indicated high-wall and open-pit side slope angles are only tentative and should be adjusted to fit the specific mining, geological and technological conditions, and also the type of roads accommodated on the benches.

Number of Blocks

With railway transport one bench has from three to five power shovels working on it (corresponding to the number of blocks), whereas with motor-truck haulage scores of them may be engaged. When the quality of the mineral is non-uniform (particularly in metal-ore quarries) the length of blocks varies widely depending upon the number of grades of the mineral.

Total Volume of the Bulk Mined Within the Quarry Outlines. This can be tentatively estimated by using the following formula proposed by V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR:

$$V = S_f H_p + 1/2 P H_p^2 \cdot \cot \gamma_{av} + \frac{\pi}{3} H_p^3 \cdot \cot^2 \gamma_{av}$$

where S_f = open-pit floor area, sq m;
H_p = open-pit depth, m;
P = perimeter of open-pit floor, m;
γ_{av} = averaged open-pit side angle, deg.

The area, outline, and perimeter of the open-pit floor are contingent upon the size and configuration of the deposit. The open-pit

floor is given a shape as close to circular as possible so as to increase the stability of the quarry sides and cut down the amount of stripped waste.

6-4. TYPES OF WORK PERFORMED IN AN OPEN-PIT AND STANDARDS FOR RESERVES IN RESPECT TO THE DEGREE OF THEIR PREPAREDNESS FOR EXTRACTION

The following kinds of work are distinguished in opencast working of mineral deposits: (1) preparation of the ground surface—felling of trees, grubbing, diversion of rivers and streams, draining of lakes and swamps, translocation of costly structures, dewatering of the

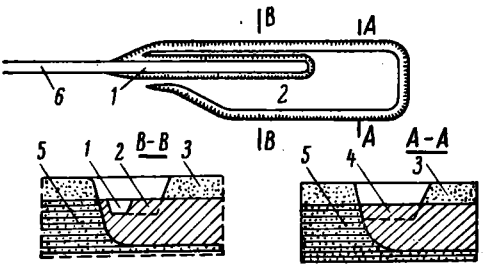


Fig. 258. Diagram for determination of opened-up and prepared reserves in a horizontal ore body with overburden:
1—working trench; *2*—prepared reserves; *3*—overburden; *4*—opened-up reserves; *5*—enclosing country rock; *6*—ingoing trench

deposit and protecting it from water inrushes, i.e. excavation of open ditches or special underground and drainage workings; (2) driving of permanent mine workings (outgoing and working trenches); (3) stripping (major development), i.e. operations such as removal of overburden and excavation of enclosing country rocks; (4) actual mining—direct extraction of the mineral from the deposit; (5) auxiliary operations—levelling out of working areas, elimination of overhangs in benches, etc.

Whenever necessary the mineral is protected from the abandoned waste (a small layer of barren rock from 0.5-1 to 2.3, usually 1-1.5 m thick is left while stripping to cut down losses of the mineral during excavation of the waste, and also when the top of the mineral is uneven). The final cleaning up is done by draglines or bulldozers.

In opencast work, as well as in underground mining, mineral reserves are classified as opened-up, prepared reserves, and those ready for extraction (Figs. 258, 259).

The opened-up reserves include the mineral contained in those sectors of the quarry benches whose top is free from the overlying

ground (or from the mineral of the superjacent bench) or is exposed owing to the natural mode of occurrence. In all these cases ingoing trenches should be driven. Areas with opened-up reserves should be pierced by the necessary drainage workings. When a bench lying below the haulage horizon is worked and there is no need to drive trenches (for example with a dragline taking ground below the elevation mark at which it operates), the opened-up reserves include the mineral lying in the portion of the bench which is either free from overlying ground or from the mineral of the superjacent bench.

Prepared reserves are reserves contained in those sectors of the quarry benches where both the top and lateral areas are exposed, thus

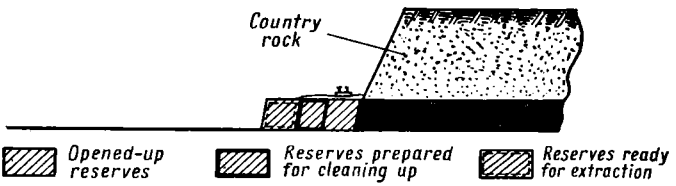


Fig. 259. Diagram for determination of reserves in respect to the degree of their preparedness in mining a horizontal coal seam

enabling actual mining to be done by the opencast method. Prepared reserves are formed by excavating the appropriate amount of country rocks in deposit areas opened up earlier.

If there is no need to take out waste the prepared reserves in the production level are formed by driving a working trench in the mineral. In order to obtain reserves ready for extraction the top of the bench is freed from the barren rock left in the sections of the deposit prepared earlier for mining (additional stripping of ground left during primary removal of overburden). In certain conditions of mining there may be no ready-for-extraction reserves whatsoever (there is no need for cleaning up). In that case prepared reserves are extracted. As in underground mining, prepared reserves form part of the opened-up reserves, and reserves ready for extraction make part of the prepared reserves.

The preparedness of reserves is established as of the 1st of April and the 1st of October each year. According to the Rules of Technical Exploitation the carry-over mineral reserves in terms of the monthly output of an open-pit should comprise:

	As of 1st of April	As of 1st of October
Opened-up reserves to last at least	6 months	12 months
Prepared reserves to last at least . .	3 months	6 months
Ready-for-extraction reserves to last at least	2 months	3 months

CHAPTER VII

CHOICE OF A METHOD FOR OPENCAST WORK, DETERMINING THE DEPTH AND PRODUCTION CAPACITY OF AN OPEN-PIT

7-1. CHOICE OF A METHOD FOR OPENCAST WORK AND DETERMINING THE DEPTH OF AN OPEN-PIT

Factors Influencing the Selection and Characteristics of Opencast Work

The major factors influencing the choice and characteristics of opencast work are: thickness of overburden and physical properties of capping and enclosing country rocks; thickness, shape, configuration and structure of the mineral deposit; its mode of occurrence (position with regard to the ground surface, angle of dip, hypsometry of bottom and top); hydrogeological conditions of mining (these should be known in order to choose a rational method of dewatering the deposit and to estimate its cost, to select the correct high-wall angles and work out measures aimed at preventing land-slides, water inrushes, etc.); feasible technical facilities for doing opencast work (by technical facilities are meant types of energy and equipment, mainly drilling, loading and transport equipment); climatic conditions prevalent in the area of mining operations (these exert a definite effect in working soft ground which requires no drilling and blasting; excavation of frozen soft ground in winter months presents some difficulties and entails considerable additional expenditures for drilling and blasting operations).

Choice of a Mining Method

In selecting the method of mining it is necessary to compare the economic efficiencies of open and underground work, except when the advantages of one of the methods are quite obvious. Failure to observe this elementary rule often leads to adverse consequences.

The basic characteristic used in economic evaluation of opencast mining is the stripping ratio, by which is generally meant the volume of excavated ground per unit of the mineral (cu m per cu m or cu m per ton).

For example $R = 2 : 1$ means that 2 cu m of waste should be removed in order to extract 1 cu m or 1 ton of the mineral. Accordingly, the value of R should be denoted as 2 cu m/cu m or 2 cu m/t.

In the practice of mining coal deposits it is customary to express the stripping ratio in terms of cu m of waste per ton of the mineral, whereas in working ore deposits this ratio is usually given in terms of cu m of waste per cu m of the mineral. The stripping ratios are classified as mean, mean exploitational, planned exploitational, overburden-to-layer, overburden-to-quarry outlines, running, and marginal (maximal) ratios.

The mean stripping ratio characterizes the average scope of stripping for the whole open-pit or a section of it.

The mean exploitational stripping ratio is the ratio of the overall volume of overburden in the quarry (minus its part that has been

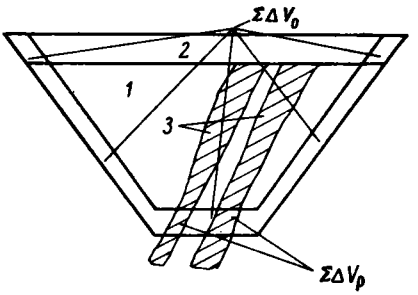


Fig. 260. Diagram for determining the overburden-to-quarry outlines ratio:
1—enclosing country rocks; 2—overburden; 3—ore body

moved to the dump pile during the construction of the open-pit) to the total reserves of the mineral less their part which was extracted during construction.

The planned exploitational stripping ratio is the ratio of the volume of the overburden to that of the useful mineral as planned for a definite period of open-pit operation.

The overburden-to-layer stripping ratio characterizes the scope of stripping operations within the bounds of the horizontal layer of the open-pit.

The overburden-to-quarry outlines ratio characterizes the relation of the volume of the overburden to that of the extracted mineral reserves added to the open-pit when expanding its horizontal contours or increasing the depth of mining (Fig. 260):

$$R_c = \frac{\Sigma \Delta V_o}{\Sigma \Delta V_p} \text{ m}^3/\text{m}^3$$

R_c is fixed when planning the operation of an open-pit in order to determine its depth, and with increase in depth it usually rises too. At the final depth the overburden-to-outlines ratio becomes equal to the marginal ratio.

The running stripping ratio is the actual relation of the volumes of the waste and the mineral excavated in the quarry over a definite period of time.

Table 30

Average Cost of Stripping and Actual Mining in an Open-Pit in Relation to the Working Conditions, roubles/m³

Mining conditions	Soft ground		Medium-hard ground		Heavy ground	
	Tonnages handled by open-pits					
	Small	Medium & Big	Small	Medium & Big	Small	Medium & Big
Excavation of ground						
Direct overcasting by power shovels under favourable conditions	0.2-0.4	0.13-0.22	0.25-0.50	0.2-0.3	0.3-0.5	0.25-0.30
Single overcasting by draglines on waste-disposal dumps	0.12-0.16	0.1-0.15	0.14-0.20	0.12-0.16	—	—
Overcasting of waste by overburden bridges	0.08-0.12	0.06-0.09	—	—	—	—
Transportation of waste in r.r. cars	0.7-0.9	0.4-0.6	0.8-1.0	0.45-0.70	1.0-1.2	0.6-1.0
Haulage of waste in motor trucks	0.6-0.8	0.5-0.8	0.8-1.0	0.6-0.9	1.0-1.4	0.8-1.2
Driving of trenches with haulage of waste	1.0-1.4	0.6-0.9	1.2-1.5	0.7-1.0	1.5-1.8	0.9-1.5
Actual mining of coal						
With bulk extraction	0.8-1.5	0.6-1.0	0.9-1.6	0.8-1.2	—	—
With separate extraction	1.0-2.0	0.7-1.2	1.0-1.6	0.9-1.5	—	—
Actual mining of ore						
With bulk extraction	1.0-1.6	0.8-1.2	1.1-1.8	1.0-1.4	1.4-1.8	1.2-1.6
With separate extraction	1.2-1.8	0.9-1.5	1.2-1.8	1.0-1.6	1.6-2.2	1.4-1.8

Note: Smaller figures reflect favourable working conditions (in mining, transportation, etc.), and larger ones refer to difficult conditions.

The marginal stripping ratio characterizes the maximum scope of stripping that is possible in opencast work.

The marginal stripping ratio expressed in terms of cu m of waste per ton of the mineral can be determined according to the formula

$$R_m = \frac{C_u - C_o}{C_e}$$

where C_u = full prime cost of 1 ton of the mineral in underground mining, roubles;

C_o = prime cost of 1 ton of the mineral in opencast mining (less expenses for removal of waste), roubles;

C_e = total expenses on excavation of 1 cu m of ground in opencast work, roubles.

The numerical values of C_o and C_e are determined in each individual case, their approximate magnitudes are given in Table 30.

The marginal stripping ratio serves for economic evaluation of opencast work.

In comparing the economic efficiencies of opencast work and underground mining the estimates of the mining cost per 1 ton of the mineral in the opencast and underground methods and of the expenses on excavating 1 cu m of waste should be made with the greatest possible accuracy, on the basis of the experience gained by similar enterprises, or on the basis of technical and economic calculations. The theoretical values of the marginal stripping ratio vary within a wide range—in the practice of mining coal deposits from 3-5 m^3/ton under adverse conditions to 10-15 m^3/ton in particularly favourable conditions of opencast mining; in working ore deposits this ratio varies from 5-6 m^3/m^3 in hard rocks to 15-20 m^3/m^3 in soft ground and with direct overcasting of ground by excavators. The smaller values of marginal stripping ratios in working ore deposits by comparison with the mining of coal beds (in cu m per ton of the mineral) are due to the difference in the volume weight of coal and ore and also to the more difficult conditions attending the exploitation of ore deposits by the opencast method (costly drilling and blasting operations, transportation of waste from the open-pit over considerable distances). It should also be remembered that the marginal stripping ratio greatly depends upon the nature and extent of mechanization of opencast work.

Determination of Opencast Work Boundaries

In working mineral deposits by the opencast method it is necessary to fix the boundaries of open work for ultimate change-over to underground mining. This kind of problem often arises in the working of steep and inclined deposits of considerable size along the dip.

Boundaries of open work can be established by the graphical or analytical procedures, the latter being less accurate since it is based on the concept of a definite geometrical position of the deposit, which rarely occurs in practice. The graphical procedure is more precise but more labour-consuming, for it requires a great deal of calculation.

The analytical method is unacceptable with deposits of extremely intricate shape, whereas the graphical method is successfully used with deposits of any shape. In order to speed up and simplify calculations the analytical procedure should be applied first to determine

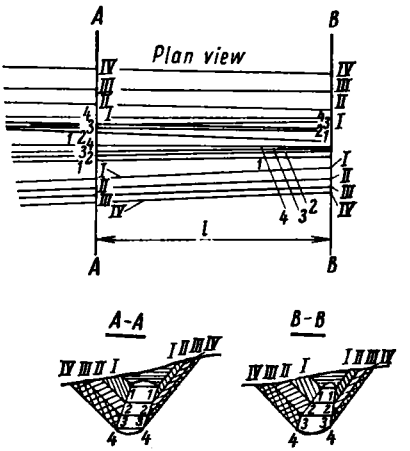


Fig. 261. Diagrams for graphical determination of the depth of an open-pit:
I-I-II-II, II-II-III-III III-III-IV-IV— cross-sectional areas of excavated country rock with reference to the corresponding horizon; *1-1-2-2, 2-2-3-3, 3-3-4-4*—cross-sectional areas of the mineral with reference to corresponding horizon

the depth of opencast work, and then adjustments should be made graphically. In this way graphical constructions may be restricted to the area lying close to the boundary of the opencast work established by the analytical method.

Graphical Determination of the Opencast Work Boundary. The essence of the method consists in the following (Fig. 261). Transverse sections along the deposit, constructed on the basis of geological and exploration data, are divided down to the depth under investigation into individual horizontal layers—benches (with due allowance for the mining conditions and equipment to be used). On breaking up the deposit into individual layers the positions of the quarry sides are plotted graphically, taking into account permissible slope angles. Then the intersection points of the quarry sides with the ground surface and the quarry floor for each transverse section are plotted on the plan of the deposit surface. The areas of country rock sections may be divided into areas with mantle and bedrock. Areas are usually measured (separately for waste and mineral) with the aid of a planimeter. The volumes of waste and

mineral for individual horizons which are contained between transverse sections are determined by the formulas:

$$V_w = \frac{F_{1w} + F_{2w}}{2} \cdot L_{AB}$$

$$V_m = \frac{F_{1m} + F_{2m}}{2} \cdot L_{AB}$$

where V_w = volume of waste per 1 m of the horizon in the area L_{AB} between section AA-BB;
V_m = volume of the mineral per 1 m of the horizon in the area L_{AB} between sections AA-BB;
F_{1w} = transverse section area in waste (section AA);
F_{2w} = ditto (section BB);
F_{1m} = transverse section area in mineral (section AA);
F_{2m} = ditto (section BB).

The volumes of waste and useful mineral for other areas of the open-pit in the 1st horizon are estimated in a similar fashion. The stripping ratio for the entire 1st horizon is determined as a volume ratio of the waste to the mineral in all areas of the 1st horizon. Overburden ratios for subsequent horizons are calculated similarly.

A comparison of the obtained overburden-to-quarry outlines ratios for individual horizons with the marginal stripping ratio possible in given conditions of mining enables one to establish the boundary of the opencast work. In doing this it should be borne in mind that the toe of the quarry may have a stepped shape (with irregular thickness of the mineral body or with considerable gangue inclusions in individual areas of the deposit).

In an open-pit of limited length (less than 1,000 m) one should also determine graphically the additional volume of country rocks which will have to be removed from the butt ends of the quarry. This volume should be added to that of the waste estimated from the transverse sections.

Analytical Methods for Determining Opencast Work Boundaries

Let us consider analytical procedures for determining opencast boundaries as suggested by some authors.

Prof. P. Gorodetsky determines the depth of an open-pit proceeding from the maximal economic efficiency, which is found as the difference between the total operating expenses in underground and opencast mining to the same depth.

The final formula proposed by P. Gorodetsky is (Fig. 262)

$$H_x = \frac{C_u - C_o}{C_b} \cdot \frac{M}{\cot \gamma + \cot \gamma'} - \frac{C_c}{C_b} \cdot h_c$$

where H_x = open-pit depth in ore, m;
C_u = cost of underground mining of 1 cu m of mineral, roubles;
C_o = cost of opencast mining of 1 cu m of mineral, roubles;
C_b = cost of excavating 1 cu m of bedrock, roubles;
M = horizontal thickness of bed, m;
γ = quarry side slope angle along hanging wall, deg;

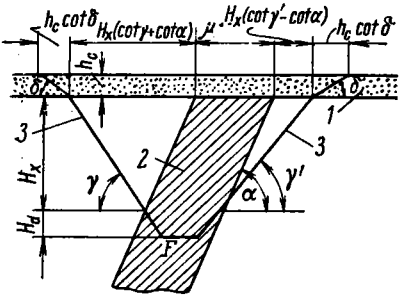


Fig. 262. Analytical determination of the pit depth:
1—cover rocks; 2—mineral; 3—open-pit bank

γ' = quarry side slope angle along foot wall, deg;
C_c = cost of stripping 1 cu m of cover rocks, roubles;
h_c = thickness of cover rocks, m.

The overall depth of the open-pit will then be (Fig. 262)

$$H = H_x + H_d + h_c$$

where H_d = deepening of open-pit without extension sideways.

Prof. B. Bogolyubov finds it inexpedient to take into account secondary factors in determining the maximum depth of an open-pit, for the initial mining and geological data serving as a background for calculations are inaccurate. The author agrees with this statement. B. Bogolyubov determines the depth of an open-pit proceeding from the equality of the marginal stripping ratio and the overburden-to-quarry outline ratio, taking into account the eventual deepening of the open-pit without extending it sideways.

B. Bogolyubov suggested the following formula for determining the depth of an open-pit:

$$H = H_x + H_d = \frac{T(C_r R_m + 1) - F}{\cot\gamma + \cot\gamma'}$$

where T = horizontal thickness of the ore body, m;
F = minimum possible width of the open-pit floor, m;
γ and γ' = quarry side slope angles;
C_r = coefficient of ore recovery in mining;
R_m = marginal stripping ratio;
H_x = open-pit depth in ore with extension sideways.

These formulas are applicable to a deposit of great length along the strike. When the length is limited $\left(\text{with } \frac{L}{H} < 5\right)$ formulas taking into account the removal of waste from the butt ends of the open-pit should be used. For such cases (in the absence of waste inclusions in the ore body) B. Bogolyubov suggested a simplified formula

$$H_x = 0.25 \tan \beta \left[\sqrt{(L+T)^2 + 4LTC_rR_m} - (L+T)\right]$$

where H_x = open-pit depth with extension sideways, m;
β = open-pit side slope angle (for simplicity all quarry side slope angles are taken to be equal), deg;
L = length of deposit along strike, m;
T = horizontal thickness of the ore body, m;
R_m = marginal stripping ratio, m^3/m^3;
C_r = coefficient of mineral recovery.

The analytical methods of P. Gorodetsky, B. Bogolyubov and other authors can be used in mining simple-shaped mineral occurrences. In working intricately-shaped deposits the method of variants is used.

Additional Information on Analytical Methods for Determining Open-Pit Parameters

In order to construct the final contours of an open-pit with a variable thickness of the ore body the quarry depth is determined for several most typical cross-sections, on the basis of which points of intersection of the quarry side lines with the top surface and floor of the open-pit are plotted in plan. Connecting these points by a smooth line we obtain the open-pit contours along the top surface and floor. The outlining of the open-pit butt end areas is done with due consideration for permissible curves for the chosen type of transport. After the top and bottom outlines of the open-pit have been constructed a longitudinal section is made, where the quarry floor usually has a stepped shape, for the open-pit depth differs from one section to another. The quarry floor is then levelled out in accordance with the nature of mineralization in the marginal horizon and the mode of transportation from lower horizons. In some cases the floor may remain benched, provided this does not affect transport operations. The plot of the open-pit in its final form makes it possible to estimate the mean stripping ratio, which should be smaller than the marginal stripping ratio. Provision should be made in the constructed open-pit for the necessary transport communication lines. In some cases, when ore bodies have a relatively regular mode of occurrence, a weighted mean section is plotted by Prof. A. Arsentyev's method on the basis of several cross-sections. In

this way it is more convenient to determine the parameters and volume of the open-pit.

The overall volume of an open-pit (Fig. 263) consists of the volume of ore V_o, the volume of two lateral trihedral prisms $V_{l.p.}$, the volume of the cover rocks overlying the ore body $V_{o.b.}$, the volume

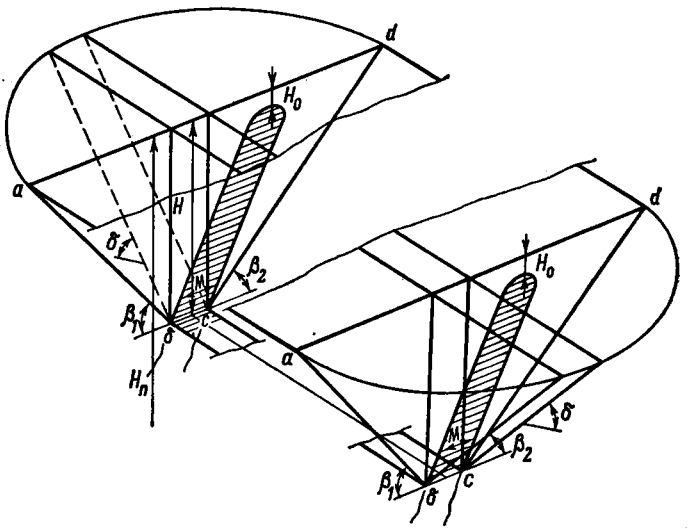


Fig. 263. Diagram for calculating volumes of ground and mineral to be excavated

of the two butt-end trihedral prisms $V_{l.p.}$, and the volumes of the cone elements V_{con}. These volumes are calculated by using the formulas given below:

$$V_o = mL(H - H_o)$$

$$V_{l.p.} = 0.5H_o^2(\cot\beta_1 + \cot\beta_2)\cdot L$$

$$V_{o.b.} = mL\cdot H_o$$

$$V_{l.p.} = H^2\cdot\cot\sigma\cdot m$$

$$V_{con} = \frac{\pi H^3}{6}(\cot\beta_1 + \cot\beta_2)\cdot\cot\delta$$

where H = depth of opencast mining, m;

H_o = thickness of cover rock over ore body, m;

L = length of deposit along strike, m;

m = weighted mean horizontal thickness of ore body, m;

β_1 and β_2 = slope angles of the left and right quarry sides at the time of reclamation, deg;

δ = slope angle of quarry butt ends, deg.

In recent years considerable improvements in the methods of defining the depth for opencast mining have been suggested. Investigations conducted by V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR, revealed that in delimiting the fields of application of open and underground work one should consider the actual schedule of the calendar distribution of volumes constructed on the basis of investigations into the conditions of mining, improve it and earmark for open work maximal reserves of a given deposit.

In determining the depth of opencast mining one should also ascertain the efficiency of utilization of capital investments, estimated costs and, in some instances, reduced profits. Prof. B. Yumatov determines the depth of opencast mining in two stages. In the first stage the initial depth of an open-pit is determined on the basis of minimal summary running expenses on the mining of a deposit by the open and underground methods. In the second stage estimated expenditures are determined for several mining variants and the most effective variant corresponding to minimal estimated expenditures is selected. The initial depth of open mining H is determined by the following formula on the basis of minimal operating expenses:

$$H_{om} = \frac{-b + \sqrt{b^2 + 2a \cdot R_m \cdot m \cdot L}}{a}, \text{ m}$$

where coef. $a = \pi (\cot \beta_1 + \cot \beta_2) \cot \delta$;
coef. $b = 2 \cot \delta \cdot m + L (\cot \beta_1 + \cot \beta_2)$;
R_m = marginal stripping ratio, m^3/m^3.

After determining the value of H_{om} several variants with a depth of 1.3-1.4 H are considered, since in many cases the minimum of estimated expenses lies within this region (as shown by investigations).

Estimated expenses E are found according to the formula:

$$E = C_a + I_i \cdot \Sigma I_r, \text{ roubles}$$

where C_a = annual working expenses, or final cost of annual production, roubles;
I_i = standard index of the efficiency of capital investment (I_i = 0.1-0.14);
ΣI_r = summary reduced investments in construction, maintenance of production capacity, and reconstruction of open-pit.

The values of C_a and ΣI_r depend upon the productive capacity of the open-pit, the time needed for its construction, expenditures per production unit, and a number of other factors. Reduction for-

mulas and methods for determining estimated expenditures may be found in the book *Tekhnologiya otkrytykh gornykh rabot pri kombinirovannoy razrabotke rudnykh mestorozhdeniy* (Technology of Opencast Mining in Combined Working of Ore Deposits) by Prof. B. Yumatov, "Nedra", 1966. V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR, has worked out a scientific procedure for investigating the conditions of mining which is used by him in constructing charts for the mean, overburden-to-quarry outlines, running and marginal stripping ratios. First, the open-pit depth is established proceeding from the actually adopted sequence of mining, provided the running stripping ratio equals the marginal stripping ratio. At this juncture the top outlines of the open-pit are fixed, this is followed by the determination of its ultimate depth, imparting to the slope angles of the quarry sides the values specified for the reclamation period. The mean stripping ratio should not exceed the marginal stripping ratio. It is also recommended to check the values obtained for the overburden-to-quarry outlines ratio.

Prof. A. Arsentyev suggested that in investigating the cycle of mining operations graphs be plotted indicating the dependence of the growing volumes of ore P on the mounting volumes of the stripped waste $V.V = f(P)$.

Based on an analysis of these graphs the running stripping ratios are averaged according to the stages of open-pit exploitation and their maximal value is found. According to A. Arsentyev the depth of an open-pit should be such that during exploitation the sum of the maximal averaged running stripping ratio and the initial stripping ratio, reduced as regards the costs to the running stripping ratio, does not exceed the value of the marginal overburden ratio (the initial stripping ratio is the ratio of the volume of the waste excavated at the expense of capital outlays to the total volume of the mineral within the bounds of the open-pit). In deducing various equations linking the mining cost with the design parameters of the open-pit it is customary to find the extremal values of the parameter sought for by equating the first derivative to zero. In all cases the range of optimal values of this parameter is fairly wide. When analysing the problem of the maximal possible depth of an open-pit M. Agoshkov, Corresponding Member of the Academy of Sciences of the USSR, came to the conclusion that in this particular case, apart from the mean prime cost, it is necessary to estimate the prime cost of additional product obtained as the result of extending the open-pit boundaries. Here the range of optimal values of the parameter sought narrows down considerably (M. Agoshkov, *Konstruirovaniye i raschety sistem i tekhnologii razrabotki rudnykh mestorozhdeniy*—Designing of Methods and Technology of Mining Ore Deposits. "Nauka", 1965, pp. 198-219).

In recent years it has been suggested that the depth of an open-pit be determined proceeding from the maximal reduced profit for the entire period of working the deposit. This criterion is the most rational economically. Using advanced methods of designing, planning bodies have prepared projects for the opencast mining of a number of deposits to a depth of 500-700 m. Under certain conditions the depth of open work for some of them may be increased to 800-900 m.

An essentially novel feature is taking into account the per-unit-capacity profit which in the final analysis may influence the choice of the mode of mining.

In all methods for determining the depth of open work and constructing the open-pit outlines the marginal stripping ratio is the most important characteristic. Its determination requires more precise data.

7-2. INITIAL DATA FOR SOLVING PROBLEMS INVOLVED IN OPENCAST MINING OF DEPOSITS

Such data should include:

detailed information on the reserves of the mineral in individual sections of the deposit with indications as to their content of useful components. The reserves must be classified into separate categories and grades of the mineral;

primary geological documents, such as field-notebooks, driller's logs and records of exploratory openings, borehole and mine-opening sections;

level-wise plans to a scale of 1 : 1,000 to be made with a spacing of 10-15 m, and 5 m in difficult mining conditions (the plans should be correlated with the bench height);

plans of boreholes with a spacing of 50-100 m, depending upon the complexity of the deposit;

longitudinal and transverse sections along the lines of boreholes to a scale of 1 : 200-1 : 500, depending upon the size of the deposit;

ground-surface plan with contour lines (spaced at 1-2 m) to a scale of 1 : 1,000-1 : 2,000;

general surface plan to a scale of 1 : 2,000-1 : 10,000;

geological map of the district to a scale of 1 : 50,000, geological maps of the deposit to a scale of 1 : 5,000-1 : 10,000 and of its individual areas to a scale of 1 : 500-1 : 2,000;

hydrogeological sections along drain drifts and exploratory lines to a scale of 1 : 2,000-1 : 5,000;

contour lines of the aquifer;

isolines of the thicknesses of the mineral and ground;

contour lines of the bottom and roof of the mineral deposit;

detailed characteristics of the physico-mechanical properties of the mineral and host rocks; stability (is considered in fixing the bench height, slope angle, and the method of working); hardness, jointing and weathering of rocks (is considered in choosing the mode of breaking off the mineral and ground from their solid mass in place); lamination of rocks (is considered in locating drill- and boreholes); water-bearing capacity of the ground (is considered in assessing the stability of the ground and in selecting the type of explosive); friability of the ground (is considered in loading and transportation of rocks); the angle of natural repose in situ and in fragmented state (is considered in excavation and loading of ground); volume weight of solid ground in place and when fragmented (is considered in loading, haulage and storage).

The necessity of timely and correct compilation of the abovementioned characteristics should be brought to the attention of the geological service staff. The accuracy of the initial data affects the correct solution of complex issues associated with the opencast mining of useful minerals.

7-3. TYPICAL EXAMPLES OF OPENCAST MINING

Proceeding from the mining and geological conditions of deposits one may single out typical examples of the use of opencast mining in working mineral deposits (Fig. 264);

1. With a flat relief of the surface a mineral deposit of sizable thickness occurs horizontally or has a slight dip and is overlain with a thin stratum of cover rocks.

Examples are Soviet deposits of limestones. A distinctive feature of opencast mining of such deposits is a more or less constant amount of overburden stripping work.

2. In inclined deposits with a dip angle closely approaching the angle of repose of bedrocks (40-60°), with outcrops situated close to the ground surface and overlain with a thin stratum of cover rocks.

3. A thick deposit with a vertical or steep pitch (60-90°) is exposed on the ground surface or covered with a very thin stratum of barren rocks, this permitting the working of the top portion of the deposit with removal of a very small amount of overburden. Examples are some chalcopyrite and chromite deposits in the Urals. In working the lower sections of these deposits the amount of waste excavation increases and therefore opencast work may be continued to a definite depth.

4. A deposit of considerable thickness is exposed on a hillside, or the hill itself consists of the mineral (mountain Magnitnaya).

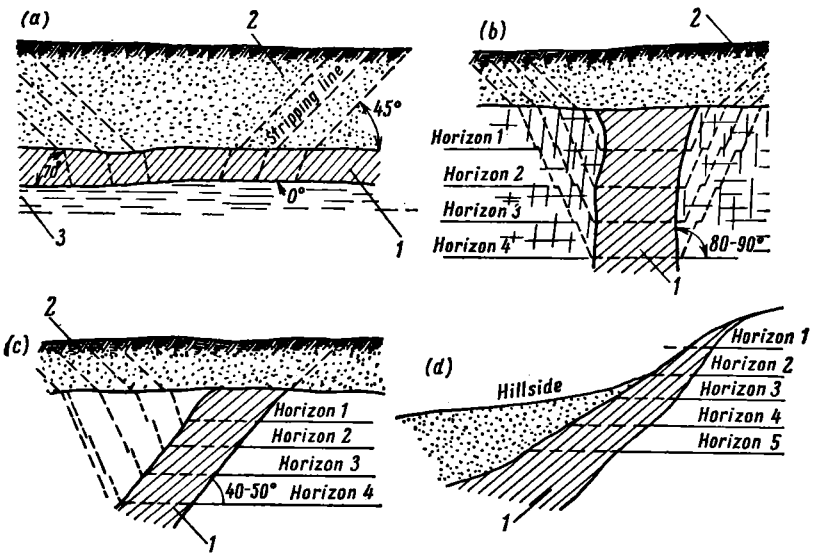


Fig. 264. Typical examples of opencast mining:
a—horizontal deposit; *b*—steeply dipping deposit; *c*—inclined deposit; *d*—deposit occurring on the hillside; *1*—mineral; *2*—cover rocks; *3*—basement rocks

7-4. DETERMINING ANNUAL OUTPUT OF AN OPEN-PIT

The output of an open-pit is measured in terms of the mineral (A_m), country rocks (A_r) and the bulk mined, i. e. the sum-total of the mineral and country rocks (A). These figures are bound by the relationship

$$A = A_m (1 + R_{r.s.}),$$
$$A_r = A_m \cdot R_{r.s.}$$

where $R_{r.s.}$ is running the stripping ratio for a given period of quarry operation.

The production capacity of an open-pit is actually determined when verifying its projected output in relation to the long-term plan for the development of the corresponding industry.

Annual Open-Pit Tonnage as a Function of Mining Conditions. The open-pit output in terms of the useful mineral can be estimated according to the formula

$$A_m = VS\gamma C_r (1 + F_d), \text{ tons/yr}$$

where V = mean annual progress in depth of opencast operations, m/yr;

S = average area of mineral within the bounds of quarry, sq m;
γ = volume weight of mineral, tons/m^3;
C_r = coefficient of mineral recovery in mining;
F_d = mineral dilution factor in mining.

The above formula is applicable in working steep and inclined deposits.

The Central State Institute for Mine Projecting recommends that in new open-pit projects the progress of stoping in depth be set at 11-16 m per year in semi-ledge and 9-14 m in hard-ledge rocks.

With progress in open-work mining the rate of advance in depth may be greatly increased and this should be considered when planning new open-pits.

The open-pit output largely depends upon the intensity of progress in depth and the rate of advance of the production benches. The limiting factor is usually the possible rate of progress in depth. By using railway transport and ЭКГ excavators this can be brought up to 15 m/yr. With motor transport the rate of progress in depth can be increased to 35-55 m/yr, as proved by actual practice at some of the Soviet open-pits.

The possible rate of progress in depth in an open-pit can be estimated by using A. Arsentyev's formula.

The annual output of an open-pit may be determined more precisely by calculation. To do this it is necessary to establish the number of production benches, active blocks, operating power shovels and their performance.

The annual volume of removed waste is

$$A_w = n_s a_w$$

Tonnage of mineral yielded by the open-pit

$$A_m = n_m a_m$$

where A_w = volume of waste removed, m^3/yr;
n_s = number of active stone benches;
a_w = volume of waste excavated in a bench, m^3/yr;
A_m = mineral output of open-pit, m^3/yr;
n_m = number of active mineral benches;
a_m = volume of mineral excavated in a bench, m^3/yr.

The number of active benches can be determined by the methods of Prof. V. Rzhevsky or Prof. A. Arsentyev on the basis of the existing mining conditions.

To find the values of a_w and a_m it is necessary to determine the number of power shovels which can be accommodated on a bench, proceeding from the minimal length of the power shovel and the annual output of each excavator.

The maximal rate of advance of the working front on an active bench can be estimated from the formula

$$l_{a.max} = \frac{Q}{hL_u}$$

where Q = power shovel capacity, m³/yr;
h = bench height, m;
L_u = minimal length of power shovel, m.

In practice the annual rate of advance of the working front varies from 50-150 to 250-400 m. An advance of up to 100-150 m is most typical for mining systems involving the haulage of waste to outside dumps (Table 31), while progress of up to 250-400 m is characteristic of transport-free methods (overcasting of waste). With benches of various length and height the volumes of the material handled on each of them depend on their respective length and height. Annual tonnages yielded by an open-pit are estimated in relation to the possible number of active benches in each year, account being taken of the mutual arrangement of individual benches and the necessary slope front angles and the size of bench working areas being observed.

When planning the annual yield of an open-pit over a period of several years one should seek to make the quarry output of ore and waste more or less uniform over a lengthy period of time (planned estimates and investigations of V. Rzhevsky and A. Arsentyev proved it advisable to level out the stripping ratio throughout the years of exploitation). A more detailed estimate of the production capacity of an open-pit, with allowance for possible deepening of the pit, the height of the bench, the effect of stripping methods, the direction of working, the means of transportation and other factors can be found in works by E. Sheshko, V. Rzhevsky, P. Zurkov, A. Arsentyev, and others.

Open-Pit Production Capacity in Relation to Economic Conditions. Projecting bodies usually stipulate the following service-life periods for open-pits:

Ore open-pits

Output of ore, mil. tons/yr	up to 1-2	2-5	5-10	over 10
Service life, yrs	15-20	20-25	25-35	40

Coal open-cuts

Output of coal, mil. tons/yr	less than 1.5	1.5-3	4.5-6
Service life, yrs	30	40-50	50-60

In very extensive mine-fields with an open-pit area of 20-30 sq. km in plan and a depth of 200-800 m, the total volume of mined bulk amounting to 2,000-10,000 mil. cu m, the service life of the open-pit is increased to 60-100 years.

Table 31

Rates of Advance of Working Front with Methods Involving Haulage of Mined Bulk (According to the Central State Institute for Mine Projecting)

Power shovel type	Category of rock hardness	Annual capacity of power shovel, thous. cu m	Bench height, m	Width of cut, m	Typical rates of advance of working front, m/yr		
					with one power shovel	with two power shovels	with three power shovels
ЭКГ-4	I-III	1,500	10	19	—	125-250	—
	IV-VIII	1,150	10-15	14-15	90-135	100-135	125-135
	IX-XIII	800			60-115	66-115	90-115
	XIV-XVI	600			46-75	50-75	66-75
ЭКГ-8	I-III	2,800	13	17.5	—	170-340	—
	IV-VI	2,200	15-20	20	112-185	122-185	146-185
	IX-XIII	1,300			66-155	72-155	86-155
	XIV-XVI	900			46-100	50-100	60-100

V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR, gives the following estimated rates for the advance of the working front in m/yr across the strike of a deposit (with power shovel loading):

Type of ground	Railway transport	Motor transport	Conveyer haulage
Soft	144-216	288-430	180-360
Dense	120-180	240-360	—
Semi-ledge rock	100-150	250-300	—
Rock	75-115	152-230	—

The above service-life periods for open-pits are based on the minimal first cost of the mineral. In open-cast mining one should compare the possible alternatives for construction of open-pits with different annual output, consideration being given to the standard rates of return on capital investment. According to the standards approved by the Academy of Sciences of the USSR, the invested capital should be completely returned within 10 years.

The obtained figures for economically expedient production capacity of an open-pit should be coordinated with the possibilities of actual mining and then adjusted in accordance with other factors.

In the working of deposits containing valuable or critical minerals the servicelife of the open pit may be reduced.

The production capacity of Soviet open pits varies from a few hundred thousand to several million tons of mineral per year. The yield of individual open-pits in terms of mined bulk amounts to tens of millions of tons per year.

CHAPTER VIII

MAJOR OPERATIONS IN OPENCAST MINING

The basic mining operations in opencast work are: drilling and blasting, loading of mined bulk at the working face and its haulage, and then disposal of overburden at waste dumps, or spoil banks.

8-1. DRILLING AND BLASTING

In the USSR and abroad electrophysical methods of breaking up the ground without the use of drilling and blasting operations are being tested. Among these methods are: the high-frequency, electric-wave, shock-wave, electrohydraulic, ultrasonic procedures, and also disintegration of rocks by applying a current of standard industrial frequency. These methods, however, are still in the stage of laboratory and field trials and hence are not considered here.

At present the drilling-and-blasting method is basically used in breaking the ground. This method is applied for breaking both the mineral and country rock. If the physical properties of the rock permit loading without preliminary loosening there is no need for drilling and blasting whatsoever.

The mineral and barren rock body in a quarry are dislodged from the solid by shooting explosive charges in ordinary holes, boreholes and, rarely, in coyote tunnels, depending upon the method of mining, the bench height and the specified lump size. The charges are usually fired in boreholes or blast-holes.

Drilling and blasting of shallow holes. As an independent method of breakage, this procedure is used relatively seldom—in low-output quarries with a limited bench height.

Holes up to 3-4 m deep and 30-32 mm in diameter are drilled with light hand-held jack hammers. In drilling holes up to 6-8 m deep with a diameter up to 50-60 mm heavy hammer drifters weighing 40-70 kg mounted on special tripods or mobile carriages are employed. The explosive charges in the holes are continuous. They are habitually fired by means of detonating fuses or electrically.

In grounds of below-average hardness (coal, chalk, marl, gypsum, soft shale) holes are drilled with electrically driven hand-held or column-mounted augers.

Drilling and blasting of boreholes. These operations are widely used in the working of minerals and grounds in open-pits of various branches of the Soviet mining industry. Boreholes on a bench are arranged in one, two and more rows.

Boreholes in grounds of average and above-average hardness are made by mobile churn-drill rigs(Fig. 265*a*), rock-roller and pneumostriking outfits, and jet-piercing (thermal drilling) units. Churn drills were widely used in the past, but are no longer manufactured, being replaced by other types of drill rigs. A brief description of churn drilling is given below.

The drilling-tool string consists of a bull bit, a drill stem and a rope socket. Simple (chisel-shaped) bits, cross bits and bits with shaped crowns are in use. Hard, compact ground is drilled with simple bits, fissured and heterogeneous rocks with cross or shaped bits. Below is the specification for the churn-drill outfit БС-1м employed for drilling boreholes in quarries.

Maximum weight of drilling-tool string, kg . . .	3,000
Operative hoisting height of tool string, m . . .	1.2
Height of mast from its base level to head sheave axle, m	14.5
Electric motor power rating, kW	60
Weight of outfit, tons	20.8
Diameter of load cable, mm	26
Borehole diameter, mm	up to 300

The drilling speed of the rig varies within a wide range, depending upon the hardness of the rocks and the diameter of boreholes (from 12 to 20 m/shift in grounds of average hardness). By eliminating down-time the drilling speed can be substantially increased. In soft ground rotary drilling outfits with screw-shaped drilling tools are employed. These units can bore holes up to 75-200 mm in diameter. Rotary rigs have the following advantages as compared to churn drills: higher drilling speed (3-4-fold); the ability of boring vertical, horizontal and inclined boreholes; lower drilling cost.

Rock roller-drill units (Fig. 265*c*) and pneumo-striking and thermal-drilling units (Fig. 265*b*) have been successfully used in the USSR in recent years. Rigs with rock-roller bits are employed for drilling in heavy ground over the entire hole bottom. In these bits the cutters are provided with cylindrical hard-metal inserts. Under a great axial pressure the cutters are pressed into the rock, slip while rotating and produce impacts on the rock, shearing off its particles.

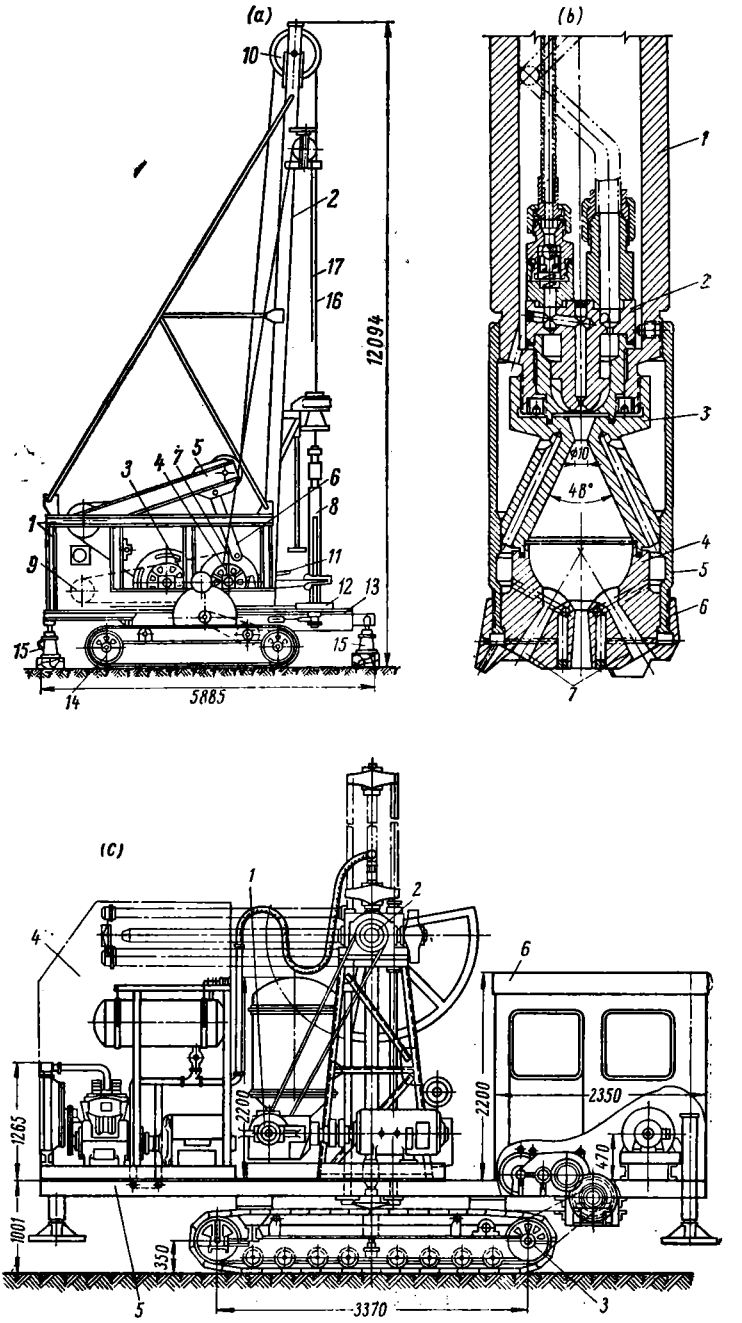
Self-propelled rock-roller units of different weight are used for drilling. They are classified as light, semi-heavy and heavy, depen-

ding upon the borehole diameter. Light units, weighing up to 30 tons, are intended for drilling blast-holes up to 214 mm in diameter, medium-heavy units weighing 45-50 tons—for boreholes of up to 250 mm, and heavy units, weighing up to 90-100 tons for 250-320 mm boreholes. Rock-roller rigs can drill blast-holes 24-30 m long at an angle of 60-90°. The pressure exerted upon the borehole bottom varies from 22 up to 70 tons, depending upon the weight of the unit and the power rating of its motor. The drilling speed of these rigs is 2-3 times as high as that developed by churn drills and reaches 20-40 m/shift in rocks of hardness 12-16, while in soft ground it may be as high as 100 m/shift. The tool string includes a three-cone bit and a set of drill stems. For drilling in heavy ground rotary-percussion units with plunging pneumatic striking pins are used. The basic features of their design have been described in the section on underground mining of ore deposits. Rotary-percussion drill rigs are self-propelled and fitted with caterpillar treads. They are used for drilling blast-holes with a diameter from 85-105 to 230-250 mm in strong rocks, the axial pressure upon the hole bottom reaching 400-700 kg. The drilling speed of a pneumostriking drill unit in open work varies from 8-10 to 25-27 m/shift, depending upon the hardness of the ground and the diameter of the blast-hole. The drill cuttings are expelled from the blast-hole by return air. The rig mechanisms impart rotary movement to the drill-rod string and ensure its continuous downward advance over its entire length (throughout the height of the bench).

Thermal drilling (jet or fusion piercing). This method consists in breaking up rocks under the effect of intense heat. The working element of the outfit is a thermal bit equipped with a reactive burner from which a gas jet at a temperature of up to 2200° C moves towards the hole bottom at a supersonic speed (up to 2,000 m/sec). A mixture of oxygen and paraffin is fed to the burner. The burner is cooled with water, which, turning into steam, carries the disintegrated slime together with combustion gases to the ground surface. Thermal drilling is used in mining very strong ores and hard rocks in the USA, Canada, the Republic of South Africa and at some open-pits in the USSR. At present thermal drilling is successfully used at

Fig. 265. Borehole drill rigs and a jet-piercing burner:

a—churn-drill: *1*—frame; *2*—mast; *3*—tool-string hoist drum; *4*—bailer hoist drum; *5*—spudding beam; *6*—crankarm; *7*—spudding beam sprocket; *8*—drilling-tool string; *9*—power plant; *10*—mast sheave; *11*—control levers; *12*—drilling tool make-and-break mechanism; *13*—platform; *14*—carrier; *15*—levelling jack; *16*—drilling cable; *17*—bailing rope; *b*—jet-piercing burner: *1*—reducer; *2*—atomizer; *3*—burner body; *4*—bottom; *5*—casing; *6*—toe; *7*—jet orifice; *c*—БСШ-1 rock-roller drilling outfit: *1*—rotor drive; *2*—rotor with feed mechanism; *3*—caterpillar tread; *4*—compressor plant; *5*—drill rig underframe; *6*—operator's cabin


(a)
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3370
3

an open-pit in the Krivoy Rog basin, where special outfits mounted on caterpillar carriages find wide application. Mechanisms for hoisting and lowering the drill stems, pump units, control desk, etc. are mounted in the rig cabin. Brief specification of the outfit:

Borehole diameter, mm	up to 160
Operating length of drill stem, m	12.5
Revolutions per minute of drill stem	8-40
Weight of unit, tons	21.6
Rated drilling speed in hard rock, m/hr	up to 4
Consumption per hour:	
of oxygen, cu m	240
of paraffin, kg	112
of water, cu m	36

A three-nozzle copper burner is connected with the drill stem through a cylindrical reducer. The latter contains pipes along which oxygen, paraffin and water are fed to the burner. In the course of drilling the burner rotates together with the drill stem. The rock

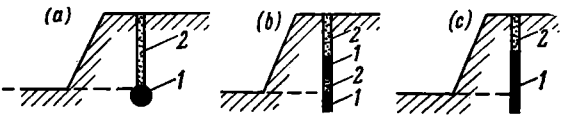


Fig. 266. Designs of blast-hole explosive charges:
a—chamber charge; *b*—divided column charge; *c*—continuous column charge; *1*—charge; *2*—stemming

disintegrated under the effect of intense heat detaches in the form of small plates or scales. Thermal drilling is effective only in strong abrasive rocks (in such grounds the drilling speed can be increased 8-10-fold by comparison with churn-drilling).

The design of thermal drilling units is being improved continuously. It has been suggested that an oxidizer (nitric acid) with compressed air be used, which will raise the speed of drilling and cut down its cost. Essentially new physical methods of well boring are being tested, the most promising ones being the hydraulic method, in which use is made of a high-pressure dense water jet spurting at an ultrasonic speed and under a pressure of over 2,500 kg/cm^2 (results of research at the Institute of Hydrodynamics of the Academy of Sciences of the USSR and the A. Skochinsky Institute of Mining) and the ultrasonic method based on the combined effect upon the rock of ultrasonic vibrations of the tools and the cavitational effect of the drilling fluid.

Designs of blast-hole explosive charges. In shooting blast- or boreholes two types of charges are commonly used (Fig. 266)—chamber

and column charges. The former are employed rather seldom (in extremely hard rocks). Their essential disadvantage is the necessity of blast-hole springing, which is a labour-consuming operation. The column charge consists of one or several columns of explosive separated by stemming. Column charges fragment the ground into smaller pieces than chamber charges do, and for this reason they find wider application in opencast work. In recent years column charges with composite explosives and an air space (Fig. 267) have

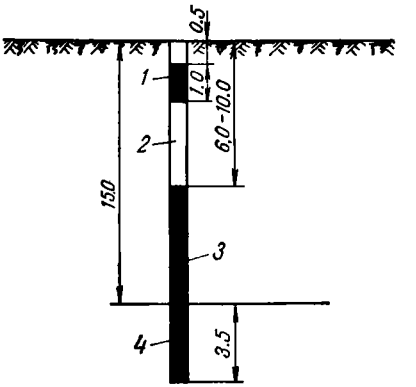


Fig. 267. Column charge with an air space and composite explosives:
1—top portion of blast-hole charged with low-strength explosive; *2*—air space; *3*—bottom portion of blast-hole charged with high-strength explosive; *4*—overdrill

been successfully used at an open-pit in the Krivoy Rog basin. Their efficiency is greater than that of ordinary column charges.

The size of the charge in blast-holes. In determining the size of explosive charges under conditions prevalent at quarries one should observe the general rules for blasting operations.

The size of an explosive charge can be found from the general formula:

$$Q = f(n)\, q \cdot w^3$$

where $f(n)$ = function of the ground blow-out index;

n = ground blow-out index $\left(n = \frac{r}{w}\right.$, where r—radius of blow-out cone and w—line of least resistance$\left.\right)$;

q = standard unit explosives consumption in kg/m^3 of ground to be blasted.

For charges yielding a normal blow-out $n = 1.0$, for a diminished blow-out $n < 1.0$, and for an increased blow-out $n > 1.0$.

In opencast work preference is given to diminished blow-out ("rending charges"), and only in driving trenches by pin-point blasting increased blow-out is used.

In open work with the use of rending charges and with a line of least resistance within the range of 5-10 m charges are calculated by the

volumetric method (according to explosives consumption per unit volume of ground to be blasted). With column charges the charge per one blast-hole in the first row can be estimated from the formula

$$Q = C \cdot a \cdot w_t \cdot H, \text{ kg},$$

where C = unit explosives consumption per 1 cu m of ground to be blasted (is determined with due consideration for blasting conditions) [1];

a = blast-hole spacing in a row (is set at 0.9 to 1.6 of w_t);

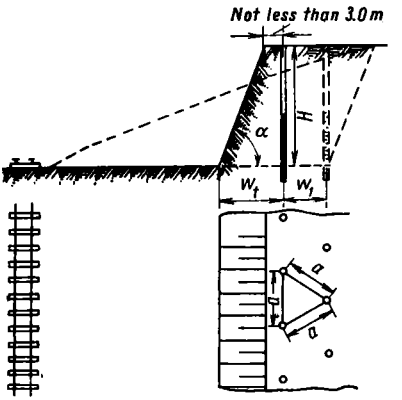


Fig. 268. Disposition of blast-holes in double-row drilling

w_t = horizontal distance from blast-hole axis to bottom of face (resistance along face toe), m;

H = bench height, m.

With high-wall angles of 50-70° the values of w_t and a can be determined by using the formula

$$w_t = H \cot \alpha + 3 \text{ m},$$

where α = high-wall angle; w_t is adopted in accordance with the strength of the ground and the bench height.

When selecting the value of w_t it should be borne in mind that blast-holes must not be placed nearer than 3 m to the bench crest.

When blast-holes are disposed in two rows (Fig. 268), they are staggered, with the spacing between rows $w_1 = \alpha \sin 60° = 0.865\ a$, m.

[1] On the average, unit consumption C with ammonite charged into blast-holes may be taken as 0.30-0.40 kg/m³ for readily blastable soft rocks, 0.40-0.50 kg/m³ for rocks of medium hardness and 0.50-0.60 kg/m³ for difficultly blastable strong rocks. In blasting very strong and tough rocks the explosives consumption may be as high as 0.8-1.0 kg/m³. More exact figures for explosives consumption are determined in each quarry for diffferent rocks experimentally.

The values of $\bar{a}$, w_t and w_1 are fixed ultimately after the completion of a series of test blasts carried out under definite conditions.

In millisecond-delay multirow blasting the size of the charge can be determined by using the formula proposed by G. Kitach, Dr. Sc. (Eng.)

$$Q = q \cdot w_t \cdot H \cdot a + 4.2q \cdot l_o^2, \text{ kg},$$

where l_o—blast-hole overdrill, m.

Many Soviet quarries have changed over from single-row blasting to millisecond-delay multirow blasting. Successful application of multirow blasting is exemplified by some open-pits of the Krivoy

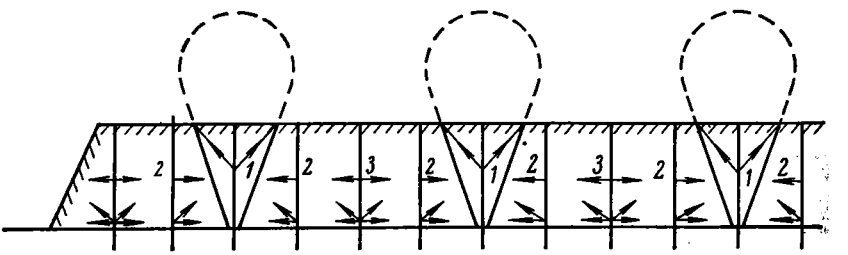


Fig. 269. Millisecond-delay multirow blasting (wave-wise pattern of shooting): *1*—cut blast-holes; *2*—blast-holes fired in a single delay-action stage; *3*—blast-holes fired in two delay-action stages; → direction of blast

Rog basin (Fig. 269). Multirow blasting enables large ready-to-load piles of ore to be built up; this improves the operating conditions of power shovels and drastically cuts down idle time of equipment because of blasting operations.

Field practice shows blast-holes of large diameter to have material advantages, such as increased yield of mined bulk per 1 meter of blast-hole. It should be remembered, however, that this increase has limits (300 mm) and in many cases it results in a higher yield of oversize material requiring extensive secondary blasting operations.

Twin bench blasting. A novel method of drilling and blasting in twin benches has been devised and put into practice on a large scale at the open-pits of the Krivoy Rog basin. Drilling and shooting of blast-holes is done to a depth of two benches at a time, the material being blasted down onto the top of the bench carrying the material from the preceding blast not yet cleaned up. The shooting is done in a tight face (Fig. 270), since this promotes fuller utilization of the energy of the explosives. Millisecond-delay blasting is used with granulated trotyl serving as explosive.

The blasted bulk is excavated benchwise. Figure 270 shows a typical case of mining with bench blasting.

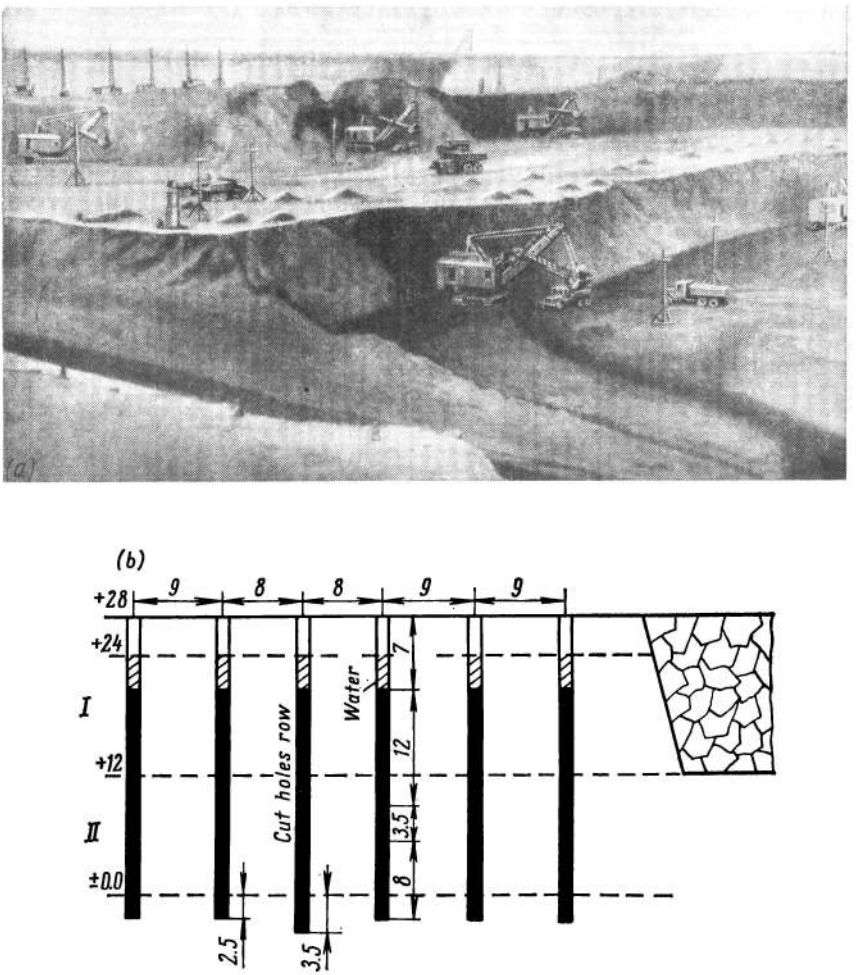


Fig. 270. Multirow blasting of twin benches at an open-pit in the Krivoy Rog basin:

a—general view of open-pit; *b*—disposition of blast-holes: *I*—first bench; *II*—second bench

The introduction of this mode of drilling and blasting has appreciably improved the breakage of ground and ore, increased the utilization of the charging volume of blast-holes, raised the yield

of mined bulk per 1 metre of blast-hole and simplified the organization of drilling, excavation and transport.

The ore yield per 1 m of blast-hole at the above-mentioned Krivoy Rog open-pit amounted to 85 cu m, twice as much as the average figure for all ore open-pits in the USSR. Fifty per cent of the broken ore have a maximum lump size of 10-15 cm, less than 15% have a maximum size of 50-70 cm.

Further improvement in multibench blasting will lead to considerable changes in the technology of mining at many open-pits, it will make possible the use of conveyer haulage and elimination of coarse crushing at concentration plants.

Explosives and blasting equipment. Ammonites are the principal explosives used for shooting blast-holes.

Dry blast-holes are charged with powder-type explosives and wet blast-holes, with cartridges having a water-proof cover (sheathed capsules).

In recent years use has been made of a mixture of granulated ammonium nitrate with liquid combustible additions—crude oil, paraffin, solar oil. Ammonium nitrate is mixed with liquid substances at the site of blast-hole charging.

Prior to charging, a plan of the disposition of blast-holes and their cross-sections should be drawn up with indications as to the traversed rocks, these being taken into account in estimating the size of charges. Blast-holes containing column charges are fired in groups of 20-30 and more. The charges are fired by using safety fuses. Two fuse bunches (basic and reserve) are used.

In the last few years millisecond-delay blasting with the aid of pyrotechnical retarders or electric delayed-action detonators (25, 50, 75, 100, 150, 250 msec) has found wide application. Millisecond-delay blasting reduces the seismic effect of the blast, cuts down explosives consumption and improves the fragmentation of the rock.

At some open-pits of the Krivoy Rog basin use is also made of combined charges: lumpy TSA (a Soviet brand of high-strength explosive) mixed with granulated alumotoluene in the bottom portion of the blast-hole, granulated powder-type explosive (granulite) in the centre, and igdanite, granulite, etc. in the top portion. In this way it is possible to use the explosives more rationally, and obtain uniform fragmentation of the ground throughout the entire bench height, thereby cutting down the cost of blasting and drilling operations.

Breakage by tunnel blasting is practised rather rarely—in high banks of up to 20 m and more, and also in driving trenches by pinpoint blasting.

In this method the charge is placed in small coyote tunnels driven from bore-pits or adits (Fig. 271). The dimensions and number of tunnels depend upon the size of the charges.

The tunnels are charged with ammonite, adding 5-10% of a stronger explosive, which is placed in the centre of the tunnel. The primer-cartridge contains several blasting caps. After the charge has

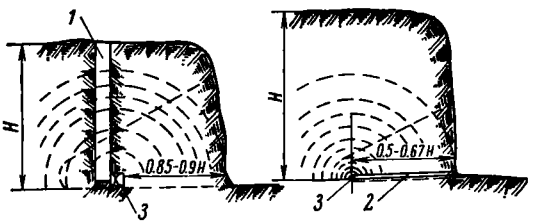


Fig. 271. Disposition of coyote tunnels in bulk caving:
1—bore-pit; *2*—gallery; *3*—coyote tunnel

been put in place all neighbouring openings are carefully filled with country rock (Fig. 272). The charges are fired by two independent methods, usually with the use of two safety fuse lines.

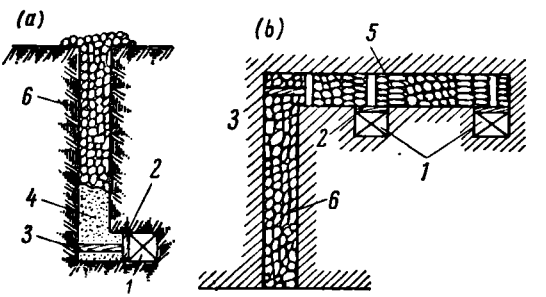


Fig. 272. Distribution of stemming and tunnel charges:
a—in bore-pit; *b*—in gallery; *1*—explosive charges; *2*—shield; *3*—stull; *4*—sand; *5*—bagged sand; *6*—rock

The shortcomings of tunnel blasting are: the large amount of work required to sink bore-pits, drive adits and charge the tunnels, and the high percentage of oversize subject to secondary breaking.

In recent years several types of machines for charging blast-holes have been designed. An example is the M3C-1 machine employed for charging blast-holes with igdanite. This is a self-propelled unit mounted on a truck chassis. The unit consists of a 4-ton capacity bin for granulated ammonium nitrate, a feeding device (reducing gear and a screw-type batcher) and a mixer. The constituents are mixed by pressurizing the ammonium nitrate by means of the solar oil supplied under pressure into the mixing chamber via injectors.

Compressed air is fed at the rate of 30 m^3/min under a pressure of 3 atm.

During blasting operations in quarries it is necessary to strictly observe the safety rules for testing, storage and use of explosives, delimitation of dangerous areas and organization of blasting.

Bulk caving with concurrent use of large amounts of explosives is an extremely responsible job which must be done according to a specially prepared technical project and with observance of all safety rules.

Secondary breaking. With properly selected parameters of the blast the yield of oversize requiring secondary breaking normally comprises maximum 3-5% of the total blasted-out bulk. The purpose of secondary blasting is to reduce the size of non-standard lumps obtained by initial blasting. In opencast mining of deposits with strong grounds secondary blasting is of great practical significance, because it facilitates loading and ensures a longer service life of costly excavating equipment. In such instances considerable attention should be given to the technique and organization of secondary blasting.

The breaking of oversize may be accomplished by the following methods:

1. Firing pop-hole or slab (unconfined) charges.
2. With a drop weight.
3. With quarry or vibrating rams.
4. Burning of thermit.
5. Applying high-frequency electric currents.

The drilling and blasting method enjoys the greatest popularity. The second and fourth procedures are inefficient. Test trials conducted by the Dnepropetrovsk Mining Institute demonstrated the high-frequency method of breaking oversize to be the most promising one. Field tests of a variant of this method proposed by the A. Skochinsky Mining Institute showed that it could raise labour efficiency per worker engaged in secondary breaking by 1.8-2 times and cut down the cost of breaking by 4-5 times with power consumption amounting to only 2-5 kW-h per 1 cu m. The production capacity of the unit is 10-15 m^3/hr. The operating principle of the installation consists in that a current from a high-frequency generator is fed to a lump of ore at two spot contacts. The ore within the contact area becomes heated and its resistance drops with the consequent development of crevices.

In the drilling-and-blasting method explosive charges are put into shallow (pop) holes drilled in a chunk of rock or placed on its outside surface (unconfined slab charges). Shallow holes are made by means of hand-held jack hammers. Blasting of oversize by using slab charges is simple but uneconomical because of high explosives

Table 32

Basic Characteristics of Drilling-and-Blasting Operations at Some Open-Pits of the USSR

Characteristics	Kounrad	Blyavinsk	Magnitogorsk	YUGOK (Krivoy Rog)	Kurgashinkansk	"Ugolny Ruchey"
Country rocks or useful mineral	Secondary quartzites, granodiorite-porphyrites	Spilites, keratophyres	Granites, porphyrites, diorites	Iron ore	Syenites, diorites	Gabbro-diabases
Coefficient of hardness *f*	10-12	6-8	8-12	10-14	9-12	12-16
Bench height, m	10-20	8-12	10	10-15	10-0	16
Blast-hole diameter, mm	230	230	250	230-300	—	250
Overdrill, m	3-4	1-2.5	2.5	2.0-3.5	2-3	4.0
Line of least resistance along toe, m	9-15	6-10	8-9	8-10	8-10	11.6
Spacing between blast-holes, m . . .	5-10	2.7-12	5-6	6.5-8	7-8	3.3
Blast-hole contiguity factor	0.55-0.67	0.4-1.2	0.75	0.8	0.8	0.28
Length of stemming column, m . .	7-11	6.8	6.0	6-7	5.5-6.5	10-12
Yield of bulk material per 1 metre of blast-hole, cu m	63.4	32.0	42.5	42	63.7	29.8
Explosive consumption, kg/m^3 . . .	0.37	0.39	0.45	0.45	0.32	0.88
Output of mined bulk per drill-rig/shift, cu m	860	495	676	250	800	100

consumption, which can be reduced by using hollow (cumulative) charges.

With unconfined slab charges the explosives consumption amounts to 1-2 kg per 1 cu m of mined bulk subject to secondary blasting, whereas in firing pop holes it decreases to 0.20-0.40 kg per 1 cu m.

Table 32 lists the basic characteristics of drilling-and-blasting operations at Soviet open-pits.

The efficiency of drilling and blasting can be raised by implementing the following measures:

(1) large-scale introduction of rotary-percussion, rock-roller and thermal drilling in working strong rocks;

(2) design and manufacture of mobile drill carriages equipped with heavy-duty drifters for drilling blast-holes up to 100 mm in diameter;

(3) introduction of multirow millisecond-delay blasting, which will increase the unit capacity of the mining front, reduce idle time for the whole mine during the blasting and the shifting of railroad, electric, compressed-air and other supply lines, and improve working conditions for excavating equipment. With multirow millisecond-delay shooting the ground can be broken off the solid mass and brought down onto the pile of rocks remaining after the preceding blast, and this makes excavation work independent of drilling and blasting. Multirow firing of blast-holes in trenches greatly increases their rate of advance;

(4) raising the efficiency of explosions by leaving air spaces between portions of the charge;

(5) placing composite charges at the bottom of the blast-hole, made up of high-loading-density or high-strength explosives (cast trotyl with addition of hexogen and aluminium), and weaker charges in its top portion. Concentration of explosion energy in the toe plane along which the ground is broken off allows the spacing of blast-holes and the yield of blasted bulk per 1 m of the blast-hole to be increased and the amount of oversize to be reduced (with millisecond-delay blasting);

(6) mechanization of blast-hole charging and stemming;

(7) further improvement of drilling and blasting parameters;

(8) supplying quarries with cheap and efficient explosives (igdanites, etc.).

8-2. EXCAVATION AND LOADING OF MUCK (ORE AND COUNTRY ROCK)

General. In working friable and soft ground excavation and loading of muck at the face are combined together, while in mining stronger rocks drilling and blasting should precede loading. Mecha-

nical methods of loading by means of excavators, scraper units and special loading machines are usually used.

Small loaders are seldom employed in open-pits and therefore are not discussed here. Scraper units are described below.

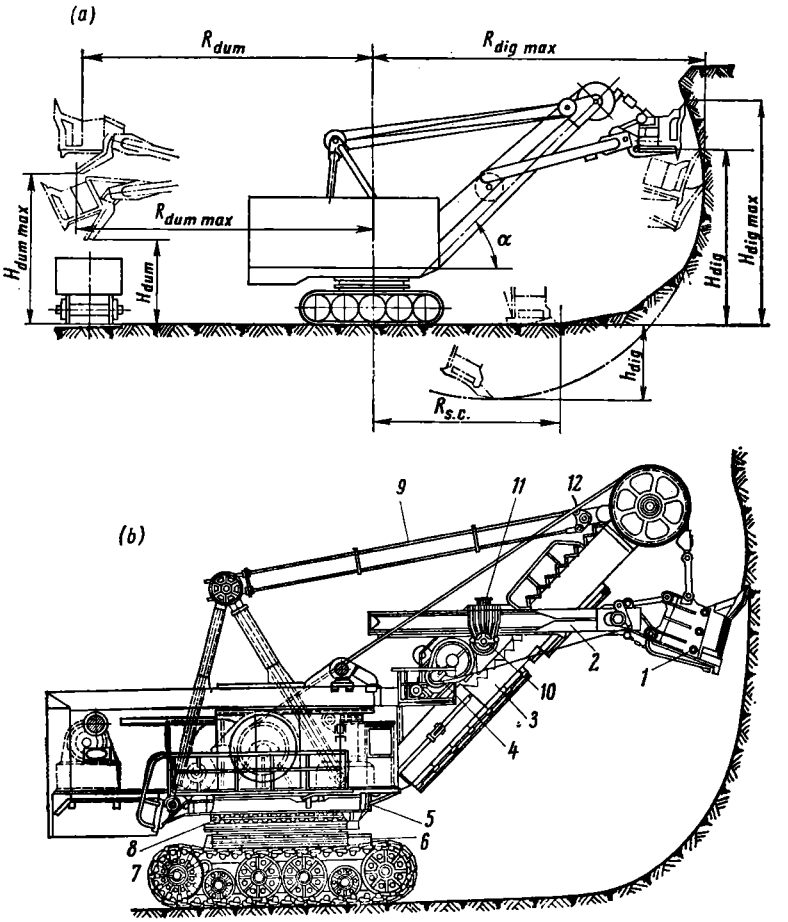


Fig. 273. A single-bucket excavator (power shovel):

a—power shovel parameters: R_{dig}—digging radius; H_{dig}—digging height; R_{dum}—dumping range; H_{dum}—dumping height; h_{dig}—depth of dig below ground; $R_{s.c.}$—swinging radius of the cabin rear; *b*—principal parts of ЭКГ-8 power shovel: *1*—dipper; *2*—handle; *3*—boom; *4*—crowding mechanism; *5*—turntable; *6*—underframe; *7*—crawler tread; *8*—bearing and slewing gear roller ring; *9*—bracing cables; *10*—crowding motion axle; *11*—saddle bearing; *12*—hoist line

The excavators used in open work are classified into single-bucket excavators (with the bucket rigidly connected with the boom,

i.e. the power shovel, and with a flexible link between the bucket and the boom, i.e. the dragline, the clamshell excavator) and multibucket excavators.

As regards the mode of their travel the excavators are subdivided into crawler-mounted, railroad-type and walking excavators.

As to their swing angle the excavators are classified as revolving (rotating through 360°) and semi-revolving (rotating through 180°).

Crawler-mounted revolving excavators are used most widely.

Today a number of Soviet machine-building plants manufacture many highly-efficient and well-designed excavators, which find wide use in the national economy.

The power shovel is the most popular type of the single-bucket excavator (Fig. 273). It consists of the operating mechanisms, turntable, underframe, carrier, and power plant.

The operating mechanisms of the power shovel include: the boom, bucket handle, bucket, hoist lines, and suspension arrangement.

The working movements of the power shovel are: crowding motion, i.e. moving the handle forward and backward; hoisting motion, i.e. moving the dipper up and down in a vertical plane; swinging motion, i.e. rotation of the whole operating equipment together with the shovel platform in a horizontal plane, and propelling motion, i.e. moving the whole excavator.

The crowding, hoisting and swinging motions of the power shovel make up one duty cycle continuously repeated in the course of excavation.

The working parameters of a single-bucket excavator are determined by using the diagram of Fig. 273.

The specifications for power shovels manufactured in the USSR are listed in Table 33.

The efficiency of a single-bucket excavator per hour of continuous work can be determined from the following formula (according to the volume of solid body in place)

$$Q_{hr} = E\eta_e \cdot n, \ \text{m}^3/\text{hr}$$

where E = bucket capacity, cu m;

n = number of duty cycles of excavator in 1 hr of continuous work at the rate of 15-40 sec per cycle (15 sec in sandy ground with small bucket capacity and 40 sec in rocky ground with large bucket capacity);

η_e = coefficient of excavation; $\eta_e = \frac{\eta_f}{k}$;

here η_f = effective bucket charge ratio equalling 0.9-0.45, depending upon hardness of ground, bucket capacity, excavator design and operator's skill. In friable and loose water-logged ground η_f reaches 1.1-1.2;

Table 33

Power Shovel Specification

Characteristics	Quarrying			Stripping			
	Э-2005	ЭКГ-4	ЭКГ-8	ЭВГ-4	ЭВГ-6	ЭВГ-15	ЭВГ-35
Bucket capacity, cu m	2.25	4-5	8-10	4-5	6-8	15	35
Swinging radius $R_{s.c.}$ of cabin rear, m . .	5.0	5.25	7.0	7.0	9.6	12.0	18.0
Boom length, m	8.6	10.5	12.0	20.5	30.0	36.0	65.0
Bucket handle length, m	6.1	7.2	8.6	12.9	20.0	19.3	44.0
Digging radius $R_{dig.}$ on ground level, m .	7.4	8.66	11.0	13.6	21.5	20.5	37.0
Maximum digging radius, $R_{dig.\,max}$, m .	12.0	14.3	17.2	22.7	35.0	40.0	65.0
Maximum digging height, $H_{dig.\,max}$, m . .	9.0	10.0	12.5	20.4	26.8	31.0	40.0
Maximum dumping height, $H_{dum.\,max}$, m	6.0	6.3	8.4	16.0	22.2	26.0	45.0
Maximum dumping radius, $R_{dum.\,max}$, m	10.6	12.6	15.3	20.9	33.0	37.8	62.0
Depth of dig below ground, H_{dig}, m . .	2.2	3.0	3.37	4.6	4.0	3.8	12.0
Power rating of drive engines, kW	100 and 75	250	520	520	520	1,450	2×1,450

Note: 1. ЭКГ-4 power shovel is at present manufactured with a bucket capacity of 4 cu m for heavy grounds and 5 cu m for soft and medium-hard grounds.
2. At present ЭКГ-4 and ЭКГ-8 power shovels are mainly used in open-pits; the latter type is employed in quarries of high production capacity.

k = expansion factor—bulking of ground in bucket, which is equal to 1.1-1.6;

η_e is commonly set at from 0.90 (sandy ground) to 0.55 (rocky ground).

The number of operating cycles should correspond to that stipulated in the excavator specification.

The efficiency of excavators during one shift in terms of cu m of solid ground in place is:

$$Q_{sh} = Q_{hr} \cdot T\eta_t,$$

where Q_{hr} = excavator output per 1 hr of continuous work;
T = duration of excavator shift, hr;
η_t = factor of excavator utilization in time (it largely depends upon the transport facilities used and the organization of empties traffic at the working face).

In practice the value of η_t is set at from 0.6 to 0.9. It may be chosen from Table 34.

Table 34

Value of the Factor of Excavator Utilization in Time (According to Regulations for Construction of Coal Open-Cuts)

Conditions of excavator operation	Type of ground			
	Soft	Compact	Semi-rocky	Rocky
Directly into waste banks (side-casting)	0.90	0.85	0.80	0.75
Loading onto hydraulic transport or belt conveyers . .	0.85	0.80	—	—
Loading onto motor vehicles	0.80	0.75	0.70	0.65
Loading onto r.r. cars	0.75	0.70	0.65	0.60

The number of working days in a year less week-ends is set at from 160 to 205, and including week-ends at from 195 to 245 (the smaller figures are for northern and the larger for southern regions).

In tentative estimates the annual output of a power shovel with loading into r.r. cars, expressed in thousands of cubic metres per 1 cu m of bucket capacity for the whole inventory stock of excavators, may be set at:

For light grounds . . . . . up to 350
For clayey grounds . . . . . up to 300
For semi-rocky grounds . . . up to 250
For rocky grounds 100-150

The annual output of a power shovel is the product of the above figures by the bucket capacity.

Dragline (Figs. 274, 275). As distinct from the power shovel, in the dragline the bucket has a flexible connection with the boom. The hoist line runs from the hoisting winch over the head pulley of the boom to the rear of the bucket. The drag rope passes from the

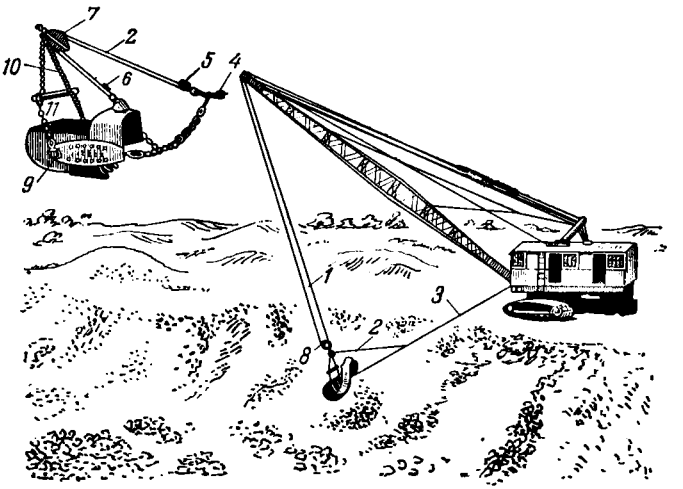


Fig. 274. Dragline:
1—hoist line; *2*—dumping line; *3*—drag rope; *4*—fixing of drag rope; *5* and *6*—fixing of dumping line; *7*—dumping line pulley; *8*—hoist line pulley; *9*—joint hinge; *10*—suspended chains; *11*—rocker arm

hauling winch via a guiding arrangement at the heel of the boom to the front end of the bucket. These lines serve to lower the bucket to the working face.

To move the bucket along the face the dragline is pulled. While moving, the bucket collects the material and is then hoisted up to the boom. Before dumping, the excavator body is swung round with the bucket pulled up to the boom to keep it from swaying.

The bucket is provided with a short dumping line. When hoisting a full bucket up to the boom it is held steady in position by pulling up the hoist line and gradually slackening the taut drag rope. When discharging the scooped up ground onto a waste bank the bucket is emptied without being lowered. In loading onto transport vehicles the hoist line lowers the bucket down to the desired height, so as to protect the vehicle from the impact of the ground discharged from the bucket.

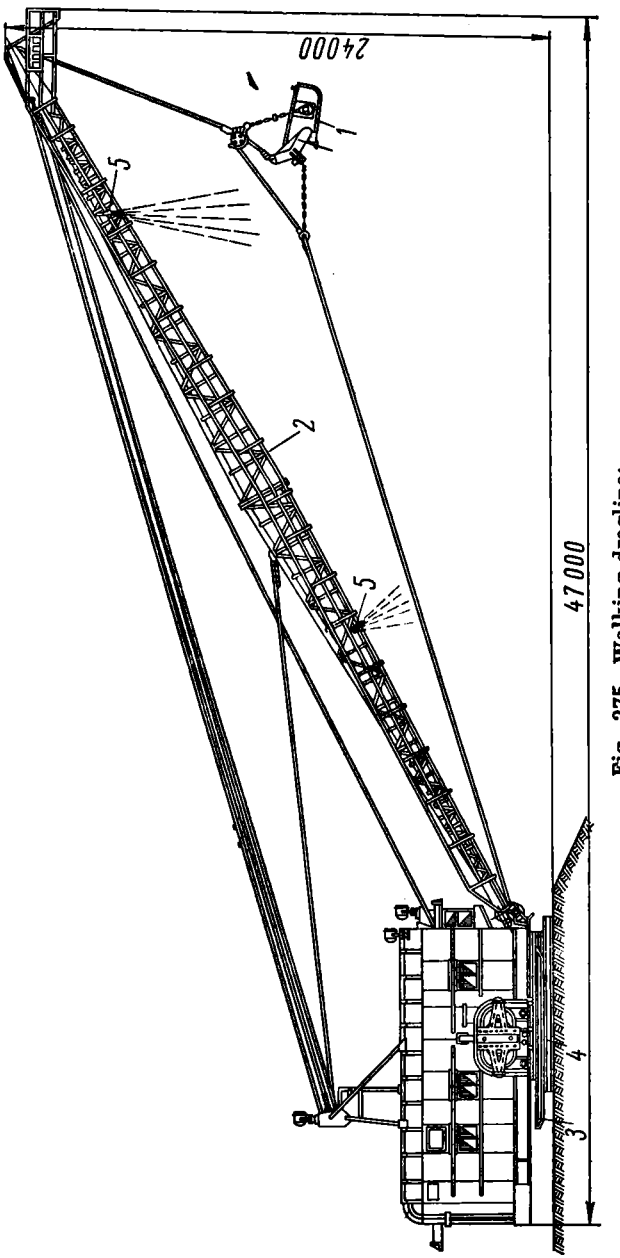


Fig. 275. Walking dragline:
1—bucket; *2*—boom; *3*—side shoes; *4*—supporting base plate; *5*—lighting

To empty the bucket the drag rope is slackened and owing to the shifting of the centre of gravity the bucket turns about its fulcrum and tips over.

The dragline carrier consists of a caterpillar truck, like that of the power shovel, or a walking mechanism (Fig. 275). This consists of two lateral supports resembling skis upon which the excavator rises somewhat and then slowly moves over a distance of 1-2 m.

Because of the low pressure exerted on a unit bearing surface the walking excavator can move over a soft ground in spite of its great weight.

The flexible link of the bucket with the boom allows it to operate only in soft grounds. Large draglines can work in moderately hard grounds as well. Scooping up of ground by a dragline is usually done at a face situated below the level from which the machine is operating. The wide range and the ability to take the ground below the level from which it operates are substantial advantages of the dragline. At the same time it is somewhat inferior to the power shovel in efficiency—with a bucket of equal capacity and in working soft ground the output of the dragline comprises approximately 80% of that of the power shovel.

Walking excavators of the dragline type with different parameters are manufactured in the USSR (Table 35).

Table 35

Specifications of Walking Excavators

Characteristics	ЭШ-6/60	ЭШ-15/90	ЭШ-25/100
Bucket capacity, cu m	6.0	15.0	25.0
Boom length, m	60.0	90.0	100.0
Dumping height H_{dum}, m	24.0	30.0	40.0
Dumping radius R_{dum}, m	57.0	81.0	95.0
Digging radius, R_{dig}, m	58.0	83.0	95.0
Digging depth, H_{dig}, m	32.0	41.0	47.0
Power rating of drive engines, kW	800	1,680	$2 \times 1,680$

The Uralmashzavod plant has designed and put into batch production a new walking excavator with a bucket capacity of 50 cu m and a boom of 125 m.

All heavy-duty walking excavators are equipped with several main drive engines (hoisting, drag, swinging and propelling) and many ancillary ones.

Multibucket (ladder and chain-and-bucket type) excavators. These machines (Fig. 276) are employed under suitable climatic conditions

in excavating and loading friable and soft ground, where no drilling and blasting is needed. Excavators of this type are used in taking out argillaceous ores, phosphorites and soft iron ores and in stripping cover rocks. The operation of multibucket excavators is usually combined with subsequent transportation of waste to spoil banks by special conveyers involving, in some instances, the use of overburden bridges and travelling stackers.

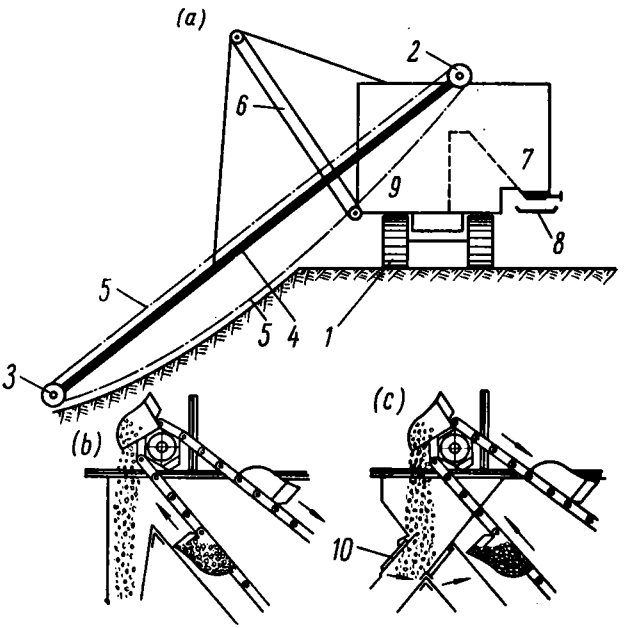


Fig. 276. Simplified diagram of a crawler-mounted down-boom chain-and-bucket excavator:

a—general operational scheme; *b*—buckets dumping their contents into receiving hopper of single gantry excavator; *c*—ditto for double-gantry excavator; *1*—crawlers; *2*—upper driving tumbler; *3*—guide tumbler; *4*—bucket chain frame; *5*—scoop chain with buckets; *6*—support boom; *7*—receiving bin with discharge hole; *8*—conveyer; *9*—operator's cabin; *10*—deflecting gate

The multibucket excavators used in open-pits may be classified into several groups according to the following features:

the design of the operating mechanism: chain, chain-and-tooth, rotary-bucket excavators;

the direction of digging: down-boom, up-boom excavators and twin excavators for simultaneous digging at the lower and upper working faces;

the design of the carrier equipment: rail-, crawler-, and motor truck-mounted excavators;

the type of power plant: with electric motors and internal combustion engines.

Below we give some information on multibucket chain excavators. These are continuously operating machines consisting of three main parts: operating mechanism, power plant, and carrier. The multibucket chain excavator strips the mineral or waste and removes it with the aid of buckets (scoops) mounted on an endless chain. The bucket chain is set in motion by a driving tumbler in the top portion of the frame. The body of a riveted excavator is mounted on one or several bogies. The stands of the excavator body carry one or several rows of motors or engines, drive gears and controls. In the body cabin or alongside it are placed receiving bins into which the mined bulk is discharged from the buckets. From the bins it is loaded into r. r. cars or onto a conveyer for further transportation.

Excavator bogies are usually equipped with independent drive engines. The bucket chain frame is of riveted design and consists of beams or lattice trusses, which carry the bucket chain. The bucket chain frame can be raised, lowered or held in a definite position with the aid of cables. The endless bucket chain passes over two tumblers—the driving tumbler, mounted in the excavator body, and the guide tumbler, located at the free end of the bucket chain frame.

The per-shift output of a multibucket excavator (in solid ground) can be determined from the formula

$$Q_{sh} = 60En\eta_e \cdot T\eta_u, \text{ cu m}$$

where E = geometrical capacity of bucket, cu m;
n = number of diggings per minute;
η_u = factor of excavator utilization in time which is set at 0.7-0.9 (0.9 when operating in conjunction with conveyers delivering waste to spoil bank);
T = hours of work per shift;
η_e = coefficient of excavation $\left(\eta_e = \frac{\eta_{ch}}{k}\right)$;
here η_{ch} = effective charge ratio of bucket;
k = expansion factor of material.

Coefficients for calculation and the number of diggings may be taken as follows:

	effective charge ratio of bucket	expansion factor	diggings per minute
Sand, gravel	0.90-1.20	1.20	30-38
Lignite, loam	0.8-1.1	1.25	22-26
Compact loamy grounds	0.75-1.00	1.25-1.30	20-24
Heavy clays	0.70-0.90	1.30-1.35	18-22

Rotary-bucket excavators. In recent years rotary-bucket excavators have found ever-increasing application in Soviet open-pits

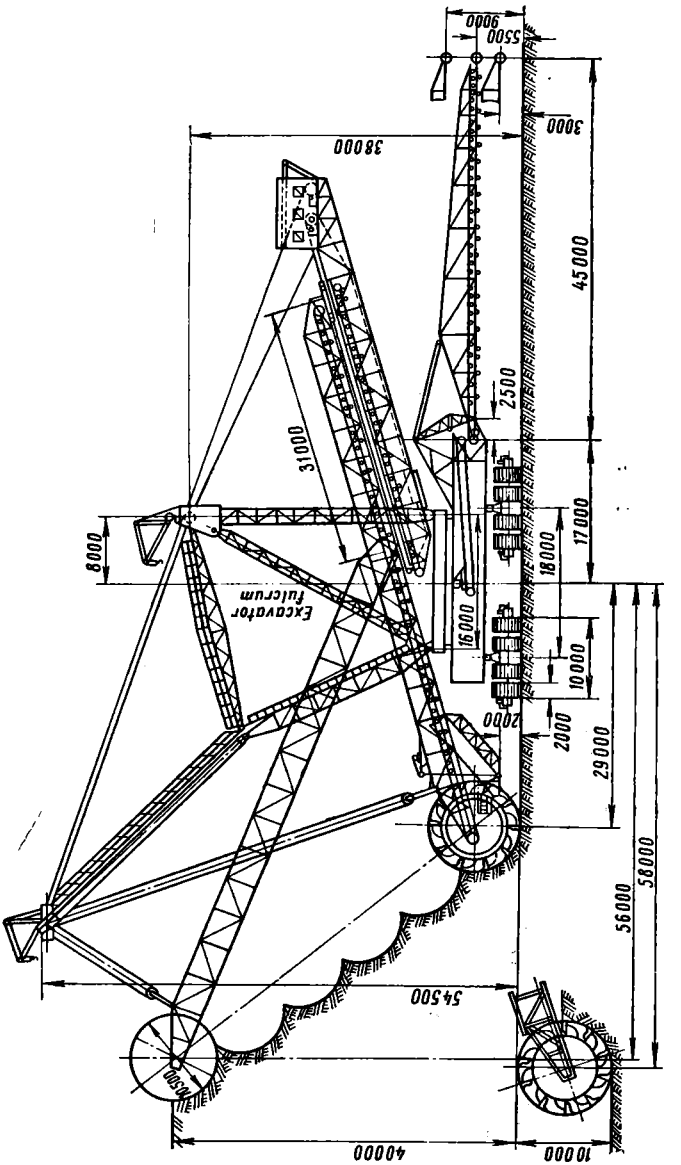


Fig. 277. Rotary-bucket excavator type ЭРГ-1,600 $\frac{40}{10}$ 31 manufactured at the Novo-Kramatorsk plant

(Fig. 277). The major structural members of this excavator are: body, boom, rotor, a set of discharge and transfer conveyers, swinging mechanism and carrier equipment.

Rotary-bucket excavators have a number of advantages as compared to chain excavators, such as a much smaller weight with a similar production capacity, a higher cutting rate (2.6-4 m/sec as against 1.0-1.4 m/sec), less wear of the rotor than for the bucket chain, better selective excavation, lower electric power consumption per 1 cu m of ground, greater shearing force at the cutting edge of the rotor buckets, which makes the rotary-bucket units suitable for work in heavier grounds.

The rotary-bucket excavator can make vertical, horizontal and combined cuts at the working face. Soviet-made excavators of this type have bucket capacities from 400 to 2,600 litres.

Rotary-bucket excavators are used in selective extraction of intricately shaped beds and in large open-pits with soft grounds. The output of rotary-bucket units attains 3,000 m^3/hr and more *in situ*. A rotary-bucket excavator with a production capacity of 11.5 thous. m^3/hr has been designed at the Novo-Kramatorsk plant. This machine, which is equipped with a mobile overburden bridge 225 m long will be the most powerful rotary-bucket excavator in the world,

8-3. TRANSPORT FACILITIES AND WASTE DISPOSAL IN OPENCAST MINING

General information on quarry transport. Transport is the major link in the technological chain of opencast work. Suffice it to say that capital outlays for quarry transport reach 70% of the total investments in open-pits, the transport operating expenses attain 50-60% and so does the labour required for transportation. The type of quarry transportation determines the mode of early development of the deposit, the method of its working and the main technical and economic characteristics of opencast operation.

Three types of transportation are most widely used today: railroad vehicles, motor trucks and conveyers. The other kinds of transport used are hydraulic, tractor-scraper and tractor-trailer transport, and cable-ways (slack-line towers), etc. According to M. Vasilyev the proportional shares of the basic types of transport used in the mining industry of the USSR are as follows:

	1966	Planned for 1970
Stripping		
Railroad vehicles	48.8	43.0
Motor trucks	35.2	36.6
Conveyers	8.2	12.9
Other types of transport	7.8	7.5

	Actual Mining	
Railroad vehicles	20.8	18.8
Motor trucks	70.1	70.7
Conveyers	2.6	1.7
Other types of transport	6.5	8.8

Railroad transport is chiefly used in open-pits with a long service life, a large bulk turnover, a relatively large size, small mining depth (about 150 m), haul distances in excess of 1.5-2 km (usually 6-7 km or more), and gradients of 3-6%, depending upon the kind of traction force used. Motor-truck transport is usually employed in working intricately shaped deposits with relatively small horizontal dimensions of the quarry considerably diminishing with depth, a moderate scope of production, haul distances from 0.2 to 5-7 km, gradients of maximum 8-10% and a road curvature radius of up to 25-30 m. Conveyer haulage is now predominantly used for transporting soft and loose ground, while the haul distances and the capacity of conveyers are actually unlimited. With preliminary crushing, conveyers can also be used for transporting rocky ground. The gradients of ordinary belt conveyers do not exceed 16-18°, in special conveyers they may reach 45-60°. In some cases it is expedient to resort to combined transport concurrently in stripping and actual mining, or during different stages of deposit exploitation (for instance, motor-truck transport in stripping on the upper horizons, and railroad transport on the lower ones).

The layout of tracks in plan and their longitudinal profile are of paramount importance in railroad transport, for here the curvature radii should be as large as possible and track gradients as small as possible. These requirements can be met in large open-pits. In this type of transport the necessary facilities include the rail track, r.r. cars and locomotives.

Rail track. This consists of the superstructure and substructure (floor) (Fig. 278). The superstructure is made up of rails, sleepers, rail bonds, and ballast. By the substructure are meant the road bed (embankments, recesses, level lengths) and ancillary structures. The track gauge depends upon the traffic. With a heavy traffic involving a throughput of over 500 thous. tons per year the standard (broad—1,524 mm) gauge is used, with smaller traffic the narrow-gauge track (750, rarely 1,000 mm) is preferred. In tracking, the design gradient should not exceed 0.015-0.020 for steam locomotive traction and 0.030-0.040 for electric locomotives. On the basis of the design up-grade the weight of the train is calculated, assuming it is pulled by one locomotive uphill at the estimated minimal speed. This is set at 10-16 km/hr for steam locomotives and at 18-25 km/hr for electric locomotives. Gradients exceeding the design value are not permissible

in open-pits. The minimum radius of curvatures is 150-200 m for standard gauge tracks and 60-100 m for narrow-gauge tracks. Additional information on the arrangement of rail tracks and turnouts is given below.

The minimum length of the horizontal section between the adverse grades is set at 150-170 m for standard-gauge tracks and 50 m for narrow-gauge ones. The minimum length of the straight line between the ends of the counter curves is 50 and 30 m, respectively. The maximum axle load is 26 and 8 tons. The width of a single-track road bed

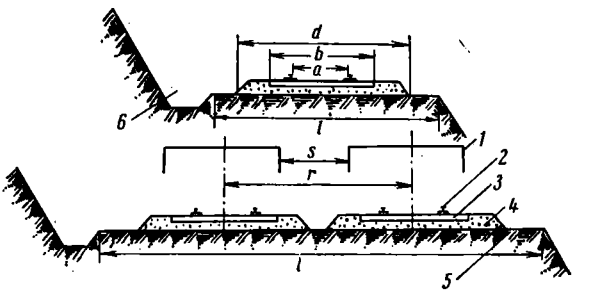


Fig. 278. Rail track arrangement in single- and double-track sections of an open-pit:

l—top width of road bed; *d*—bottom width of ballast layer (retaining prism); *a*—track gauge; *r*—track spacing; *s*—clearance between vehicles; *b*—sleeper length; *1*—rolling stock overall dimensions; *2*—rail; *3*—sleeper; *4*—ballast; *5*—road bed; *6*—side-ditch

is taken as 4.6-5 and 3.2-3.6 m, and that of a double-track road bed as 8.7-9.1 and 6.6-7.0 m, depending upon the type of ground. The mean slope of the road bed is 1 : 1.5. The ballast layer for a single-track road bed is 2.9 m wide at the top for the standard gauge and 2.1 m for the narrow gauge, its thickness being 0.15-0.35 m, depending upon the axle load and the type of ground making up the floor of the bed. A total of 1,440-1,840 sleepers are needed to lay 1 km of track. The rails weigh 38-65 kg/m in standard-gauge and 18-24 kg/m in narrow-gauge tracks.

The turnout types (tangent of the frog angle expressed as a fraction) are 1/7, 1/9, 1/11 for the standard gauge, and 1/5, 1/6, 1/7 for the narrow gauge (Fig. 279).

The permissible speed for trains is up to 40 km/hr on permanent quarry tracks and 15-20 km/hr on sectional tracks at the working faces and at waste dumps.

Rail tracks are often shifted both in the open-pit itself and at waste dumps. In metalliferous open-pits they are ordinarily shifted in separate sections by mobile cranes.

When operations are performed by means of multibucket excavators and overburden bridges, tracks are moved with the aid

of continuous-action track-shifters. At waste banks they are moved by cranes or intermittent-action track-shifters.

Mine cars. In selecting the construction of r. r. cars it is necessary to take into account the mode of loading, volume weight and hard-

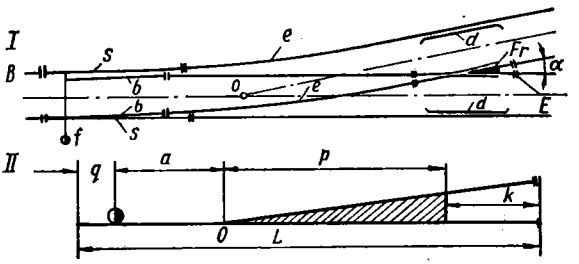


Fig. 279. Layout and basic dimensions of a r.r. turnout in plan view:
B—beginning of turnout; *E*—end of turnout; *I*—layout; *II*—basic dimensions; *s*—stock rail; *b*—point rail; *e*—adjustable curve; *d*—guide rail; *Fr*—frog; *f*—point-rail shifting mechanism; *O*—centre point of turnout (track axes intersection point); α—frog angle

ness of the ore or country rock, and the production capacity of the open-pit.

Soft and loose ground is carried in light-weight cars. In low-output quarries narrow-gauge low-capacity cars are used, and vice versa.

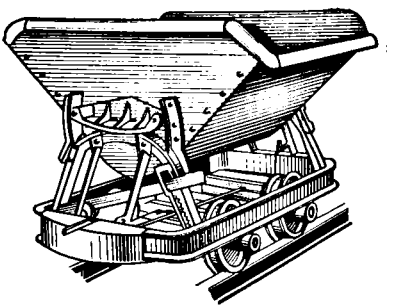


Fig. 280. 1-2.5 cu m-capacity rocker side-dumping mine car

The capacity of the car should be 4 or 5 times that of the excavator bucket.

The following types of cars are employed most frequently: 1.0-5.0 cu m cars on narrow gauge tracks (Figs. 280, 281), and 40-100 ton air-dump and gondola cars on standard-gauge tracks (Figs. 282, 283).

Mine-cars of 1-2.5 cu m capacity are discharged by tipping. The unloading mechanism in mine-cars of 2.5-5 cu m capacity is actuated by means of levers.

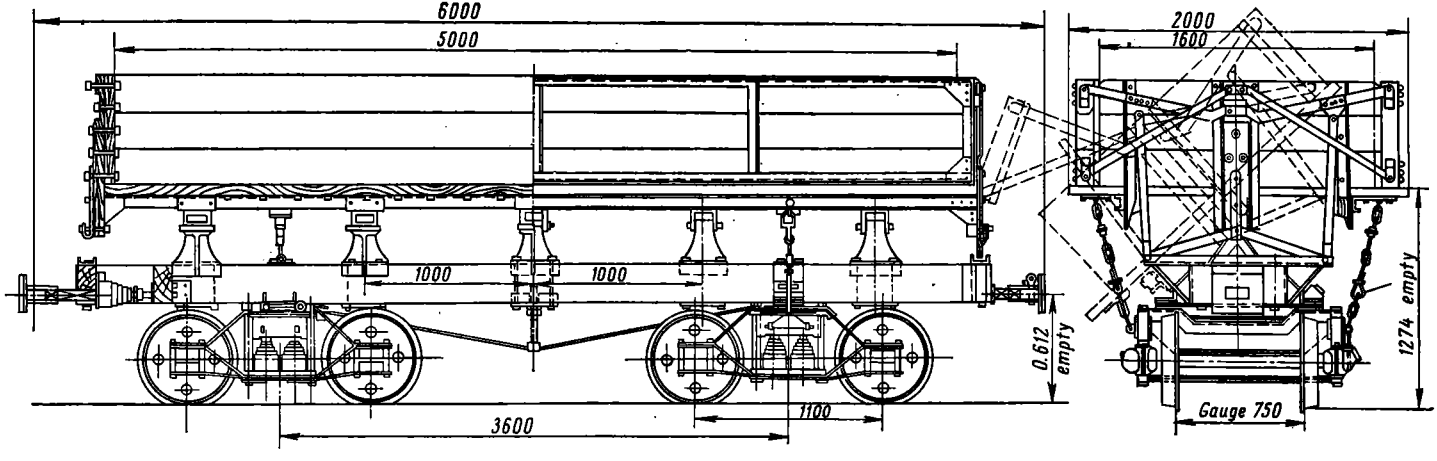


Fig. 281. 5 cu m-capacity side-tipping car dumped with the aid of levers

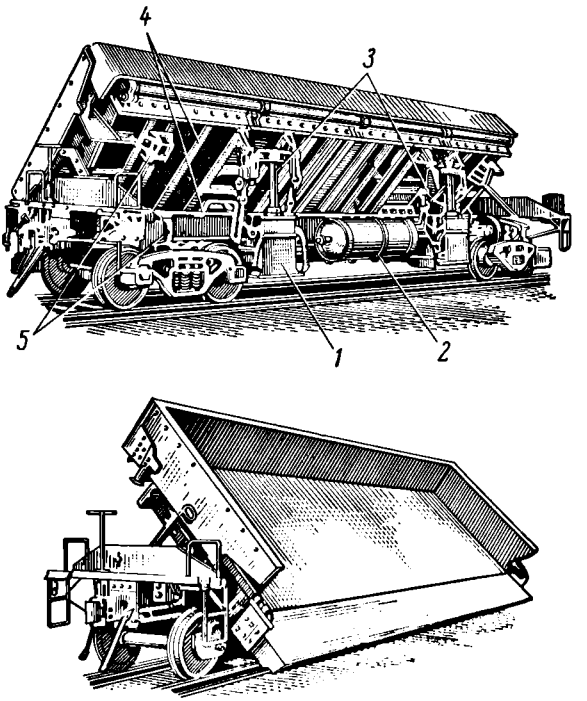


Fig. 282. Air-dump car:
1—dumping cylinder; *2*—air tank; *3*—dumping mechanism (piston and rod); *4*—stringer and cross beams; *5*—body dumping trunnions and bracket

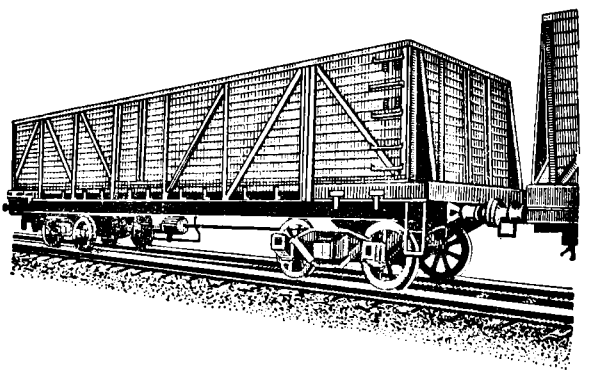


Fig. 283. Four-axle gondola car of 60 ton load-carrying capacity

Dump cars with a tipping body and a capacity of 20 tons travelling on 750 mm-gauge tracks and 50-120-ton cars on 1,524 mm-gauge tracks are ordinarily used for hauling waste to spoil banks. The body of the dump car is tipped by a pneumatic device, which is either centrally controlled—from the locomotive, or each dump car is tipped separately. The locomotive has a compressor plant supplying air at a pressure of 4-5 atm. Compressed air is fed along pipelines to the air tanks of each car. Dump cars are provided with one or two air cylinders on each side. When the car body is tipped the compressed air enters the bottom portion of the air cylinder, actuating a piston rod connected with the bottom of the car body.

60- to 125-ton cars with a solid-end metal body, or with hatches in the bottom of the body (gondolas) are used for transporting minerals. Gondola cars can be unloaded by using a revolving tipper. The standard-gauge rolling stock now in use in the USSR is, as a rule, equipped with automatic couplings and automatic brakes, which ensures effective utilization of transport facilities and safe traffic in open-pits.

To prevent freezing of the loaded ground to internal parts of the car body these are sprayed with a 3-4% solution of common salt when the temperature is —30°C or higher or with a calcium or magnesium chloride solution at temperatures down to —35 or —40°C. To avoid the freezing of the car body it is heated with gas, steam or electricity. Good results have also been obtained by applying solar oil to the internal surfaces of the car body.

Locomotives. The type of traction power and locomotives is established in accordance with the service life, production capacity and depth of the open-pit. Three basic types of locomotives are employed in open work: steam-, electric- and diesel-driven. Today steam locomotives are used chiefly in old, low-output or short-life open-pits.

In open work with dense traffic and difficult road profile trolley locomotives are used (Fig. 284). These are driven by electric motors located between the axles. A current flowing from the substation along the wires is fed via a collector (usually pantograph) to the motors, and from these—through the wheels and rails—it flows back to the substation. To reduce resistance at joints use is made of copper bond wires fixed to the rails, with occasional addition of a return cable.

The trolley wire is suspended over the haulage tracks on special upright supports. The electric loco is fed with direct current from traction substations which convert alternating current most commonly used at mines. Electric locos develop a great tractive effort at starting, negotiate steep grades and have an overall efficiency of 14-16%.

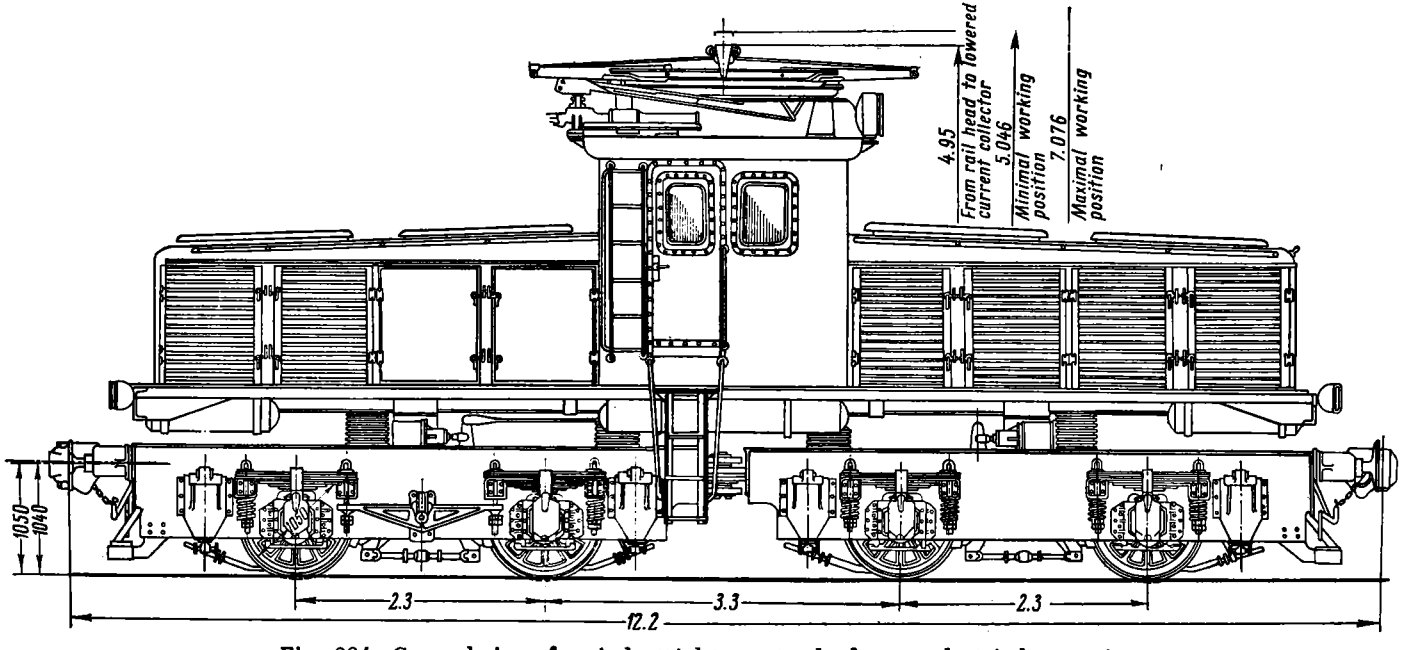


Fig. 284. General view of an industrial-type standard-gauge electric locomotive

Below we give specifications for standard-gauge electric locomotives widely used in Soviet open-pits:

Characteristics	
Adhesion weight, tons	80-150
Voltage, V	1,100-1,650
Tractive effort at one-hour power rating, kg	12,000-22,000
Speed, km/hr	23.4-24.7
Minimum curvature radius, m	40-60

The major drawback of trolley locomotives is the difficulty of maintaining the trolley network in good repair, especially during drilling, blasting and excavation. This disadvantage is eliminated in diesel-locos, which carry their own diesel-driven electric power plant. Their efficiency attains 25-28%. Diesel-locos have a number of substantial advantages over trolley locomotives.

Gasoline locomotives are fitted with internal-combustion engines and their tractive effort is usually rather low. Therefore their use in open-pits is limited to ancillary operations.

The basic problems of rail transport (determination of tractive effort, weight, travelling time, electric power consumption, etc.) are solved by calculation (see textbooks on mine transport).

The major steps to improve the operation of the open-pit rail transport system should include:

utilization (in large open-pits) of electric locomotives with an adhesion weight of up to 150-180 tons;

feeding electric locos with high-voltage alternating current;

introduction of diesel-electric locomotives, thereby completely dispensing with the trolley network;

utilization of self-propelled dump cars for operation at deep-lying horizons with road upgrades reaching 70-80‰. This will greatly reduce capital outlays and running expenses;

extensive application of high-quality light steels and alloys in the manufacture of cars.

Motor transport. *General.* With this type of transport the mineral and barren rocks are hauled in motor dump-trucks, articulated six-wheelers, and truck-type tractors with a trailer (Fig. 285). Motor transport is widely used in open-pits of the USSR.

The main advantages of the motor transport are: (1) the ability of negotiating steep gradients and small-radius curvatures (up-grades up to 0.080-0.100, minimal road curvature radius 30 m on main

roads and 10-15 m on temporary ones); large up- and down-grades and small-radius curves greatly reduce the scope of stone work, particularly in mining small and irregularly occurring ore bodies; (2) simple organization of traffic and independent operation of

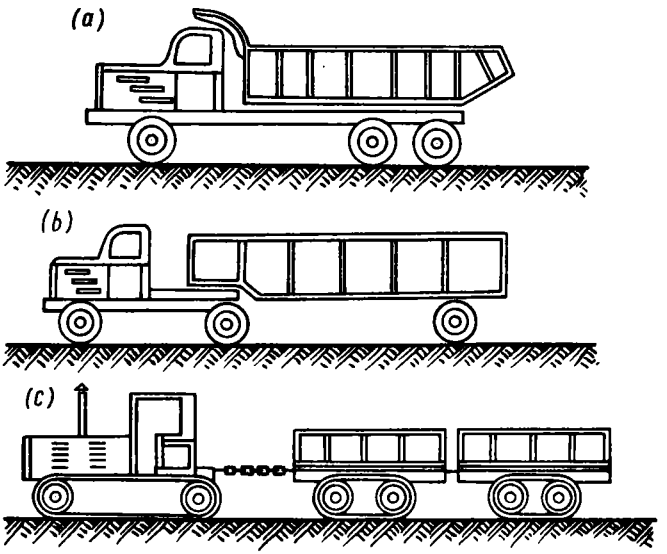


Fig. 285. Motor vehicles:
a—quarry-type motor truck; b—articulated six-wheeler; c—truck-type tractor with a trailer

individual trucks; (3) high manoeuvrability of the vehicles; (4) less time needed for the construction of open-pits; (5) better utilization of excavators in time.

Characteristics of the equipment. Diesel-driven rear dump-body motor trucks (Fig. 286) are widely used in open-pits. Motor truck specifications are given in Table 36.

Since 1961-1962 articulated six-wheelers manufactured by the Byelorussian and Kremenchug motor car plants have become very popular. The load-carrying capacity of single-axle articulated trailers varies from 30 to 65 tons. A double-axle articulated eight-wheeler with a load-lifting capacity of 110 tons (БелАЗ-549В) is slated for production. The maximum road speed of articulated six-wheelers varies from 30 to 50 km/hr. Crawler-type tractors with trailers are used for hauling two or several rubber-tired or caterpillar trailers. Trailer-tractors in use have a rated power from 60 to 250 h. p. with a road speed from 3 to 30 km/hr. Trailers with a load-carrying capacity of 10-40 tons are of the bottom-dump type. Tractors with

Table 36

Manufacturer and model of dump truck	Load-carrying capacity, tons	Unladen weight, tons	Maximum speed, km/hr	Capacity of truck body, cu m	Fuel consumption per 100 km, litres	Power rating of motor, h.p.
Kremenchug Plant						
КрАЗ-222	10	12.2	47	8	65	180
КрАЗ-254	10	11.5	60	8	55	180
КрАЗ-256	10	11.4	62	8	55	240
КрАЗ-251	11	10.55	65	6-8	55	240
Byelorussian Plant						
МАЗ-525	25	22	30	14.3	160	300
МАЗ-530	40	38.5	30	22.0	—	450
Бел-АЗ-540	27	21.0	52	15.2	—	360
Бел-АЗ-548	40	26.5	52	22.3	—	520
Бел-АЗ-549	65	45.0	47	32.5-36	—	2 × 360

trailers are employed in small-scale open work involving short-distance trips (300-400 m) over poor roads.

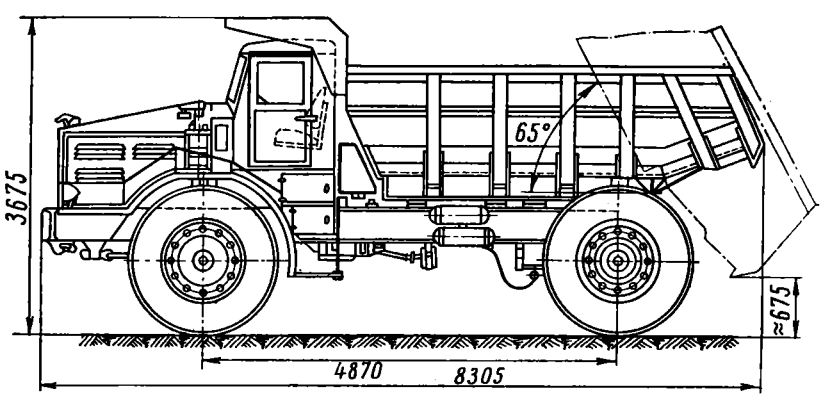


Fig. 286. МАЗ-525 rear dump-body motor truck of 25 ton load-carrying capacity

A variety of motor vehicles are trolley cars—dump-body trucks equipped with an electric motor and with power supplied from trolley wires carrying a direct current. Their main advantages lie in the simplicity of control, reliability in operation and lower

cost of transportation (about 35% less than with dump-body motor trucks).

Motor roads. The efficiency of motor transport is largely dependent upon the condition of roads and for this reason their design and maintenance should be given much attention.

In estimating the width of the roadway consideration should be given to permissible clearances between two meeting trucks and to the distance from each of them to the way-side. The width of the roadway in a single-lane road is set at 3.5 m, and in a double-lane

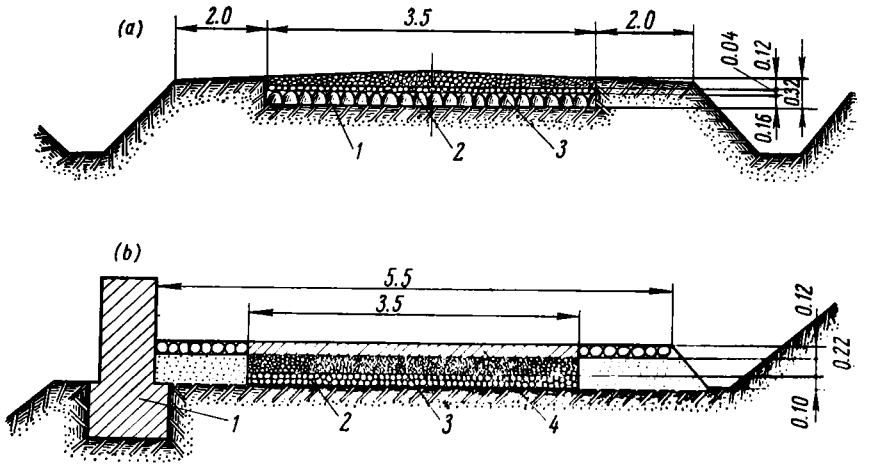


Fig. 287. Cross-sections of motor roads in open-pits:
a—at Bogurayevsk open-pit: *1*—crushed stone pavement, 25-40 mm thick; *2*—crushed stone, 40-50 mm thick; *3*—subjacent stone layer; *b*—at the Balaklava open-pit: *1*—balustrade; *2*—subjacent layer of coarse crushed stone; *3*—crushed stone, 35-45 mm thick; *4*—cement-bound crushed stone, 15-75 mm thick

road at 7-8 m. The road pavement in open-pits should be at least 0.15-0.20 m thick and consist of crushed stone, gravel or slag (Fig. 287). The quality of road pavement substantially affects the specific tractive resistance of motor cars (kg/ton of the overall weight of the machine), which is 10-11 for asphalt and concrete, 40-50 for a cobble-stone paved road, 100 for a sand road, etc.

In open-pits ballast-roads are commonly used. In recent years pre-fabricated reinforced concrete slabs have found wide application as pavement for quarry roads. The service life of this pavement is 10-15 times as long as that of ballast.

Permissible load-carrying capacity. The permissible load-carrying capacity of a motor truck is estimated on the basis of the tangential tractive effort and running resistance of the motor truck (with due

consideration for the road pavement and ruling gradient) in kilograms per 1 ton of the total weight of the truck. The dead weight of the truck should also be taken into account. The maximum permissible load-carrying capacity of a truck is verified against its adhesion weight by a special calculation procedure.

To level out the waste on dumps, to grade the sites for roads and to clean up the ground on quarry banks, bulldozers (caterpillar tractors equipped with a share) are used (Fig. 288).

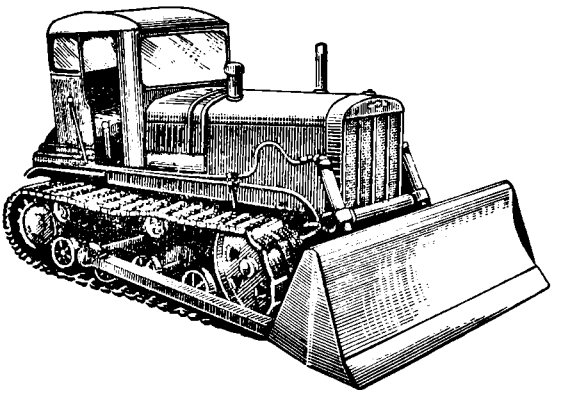


Fig. 288. Bulldozer

Conveyer haulage. Conveyer haulage is used as an independent means of transport at a number of Soviet coal open-cuts, metalliferous open-pits, and also many building-material quarries.

The advantages of conveyer haulage are: continuous delivery of the mineral, more efficient utilization of excavating equipment, low initial capital outlays, small service staff, the possibility of rock picking and of using considerable gradient angles: up to 18-20° with usual belts and up to 36° when the material is pressed against the belt with a flat-link chain netting. Hoisting stationary conveyer plants with a held-down belt can lift loads at a slope angle of up to 40-60°. The width of the belt in such conveyers reaches 2,500 mm and its travel speed, 4 m/sec. In quarries, rubber belts 650-1,400 mm wide consisting of 3 to 12 interlayers are commonly employed. At low temperatures frost-resisting rubber belts are used. High-resistant belts are manufactured with interlayers of capron, nylon, etc. Recently, belts with metal warp have been in use, and so have cable-belt conveyers.

Belts with metal warp are very durable. They consist of steel wires up to 4 mm in diameter, cotton lining, and rubber filling.

In cable-belt conveyers large-diameter steel ropes pull a light belt carrying the load.

Major advantages of these conveyers are the comparatively low cost of belting and the possibility of transmitting large tractive efforts. The length of the belting may be extended to 2.0-2.5 km and more.

A substantial shortcoming of conveyers is the difficulty of their operation in winter and rainy seasons.

The general pattern of moving the material by conveyers is as follows. The ore or waste is loaded at the face by an excavator through a charging hopper fitted with a grid or a feeder onto a short face conveyer, from which it is delivered onto a gathering transporter and then the main hoisting transporter set up on one of the quarry sides. The hoisting conveyer commonly delivers the ore directly to storage bins.

The overall length of belting at big open-pits reaches 10-12 km. Portable or stationary crushers may be set up in open-pits, if necessary. Calculation of conveyer haulage is discussed in manuals on mine transport.

Among the latest improved constructions of Soviet-made conveyers mention should be made of the KPУ-900 model with a throughput of 900 tons/hr provided with a steel-cable-reinforced belt 1,200 mm wide. Conveyer haulage, which has a number of considerable advantages over other types of transport, will certainly find wide application in the future.

Belt conveyers enjoy wide popularity as an auxiliary means of transportation in operating together with overburden dumpers and bridges.

Scraper haulage. In open-cast work use is made of power drag and wheel scrapers. The former are rarely used and therefore are not considered here.

Tractor-drawn wheel scrapers (Fig. 289). These can be used efficiently in removing overburden, driving trenches and carrying out some other secondary earth-moving and digging operations over limited haul distances from 100-200 to 1,000-1,200 m (usually up to 500 m) and with small amounts of material handled—up to 1,500-2,000 cu m per shift. A wheeled scraper is pulled either by a caterpillar tractor or by a wheel tractor. In a soft and loose ground scrapers operate without any preliminary ripping, while a firm ground has to be previously loosened up by ploughs or rippers.

The work of a wheeled scraper consists of the following operations: filling of the sraper bucket (Fig. 290) when the unit is moving and taking a cut of the ground 1-3 m wide, depending upon the struck capacity of the scraper, and 0.1-0.3 m thick, depending on the hardness of the ground; the travel of the filled scraper from the site of

loading to that of waste disposal (loaded trip); discharging of the scraper bucket at the waste bank; the travel of the empty scraper from the waste bank to the working face (empty trip).

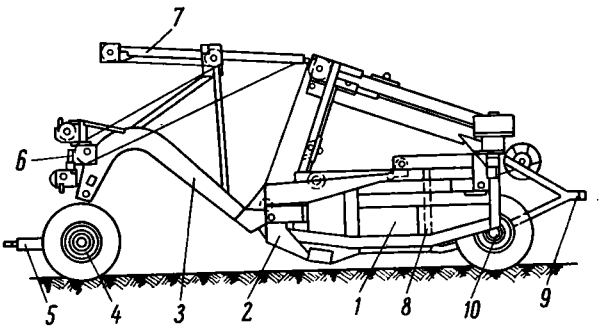


Fig. 289. Two-axle scraper:
1—bucket; *2*—front flap or apron; *3*—bail; *4*—front axle with leading wheels; *5*—traction shaft; *6*—guide idlers; *7*—beam; *8*—back plate; *9*—bumper; *10*—rear axle with wheels

When the scraper is loaded and discharged its front flap is raised. When moving over the dumped waste the scraper levels it out with

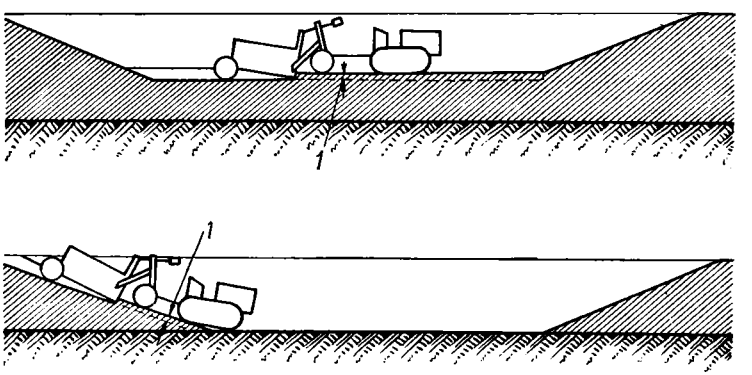


Fig. 290. Wheel scraper operation in an open-pit:
1—thickness of ground cut

the bottom plate of the bucket. When travelling with a loaded or empty bucket the scraper is in a horizontal position.

The performance of the wheeled scraper largely depends upon the capacity of its bucket, the haul distance, the nature of the ground, and the road profile. Loading is accomplished over a stretch of 10-40 m at a speed of 0.7-0.8 m/sec; the dumping time is 0.25-0.5 min;

the travel time depends on the haul distance and the travel speed, which may be taken to equal the ordinary speed of pulling tractors, i.e. 0.8-1.0 m/sec. The seasonal efficiency of wheeled scrapers over a period of 200-210 working days with a haul distance of 200-300 m is:

Scraper model	Д-147	Д-222	Д-213	Д-188
Bucket capacity, cu m	6	6.5	10	15
Volume handled, thous. cu m	75-105	80-115	120-175	160-230

When a scraper is pulled by a motor truck along a ramp road the upgrade in the direction of empty travel should not exceed 15% and in the direction of load travel 8%. With tractor-scraper haulage the slope may be 1.5-2 times as large.

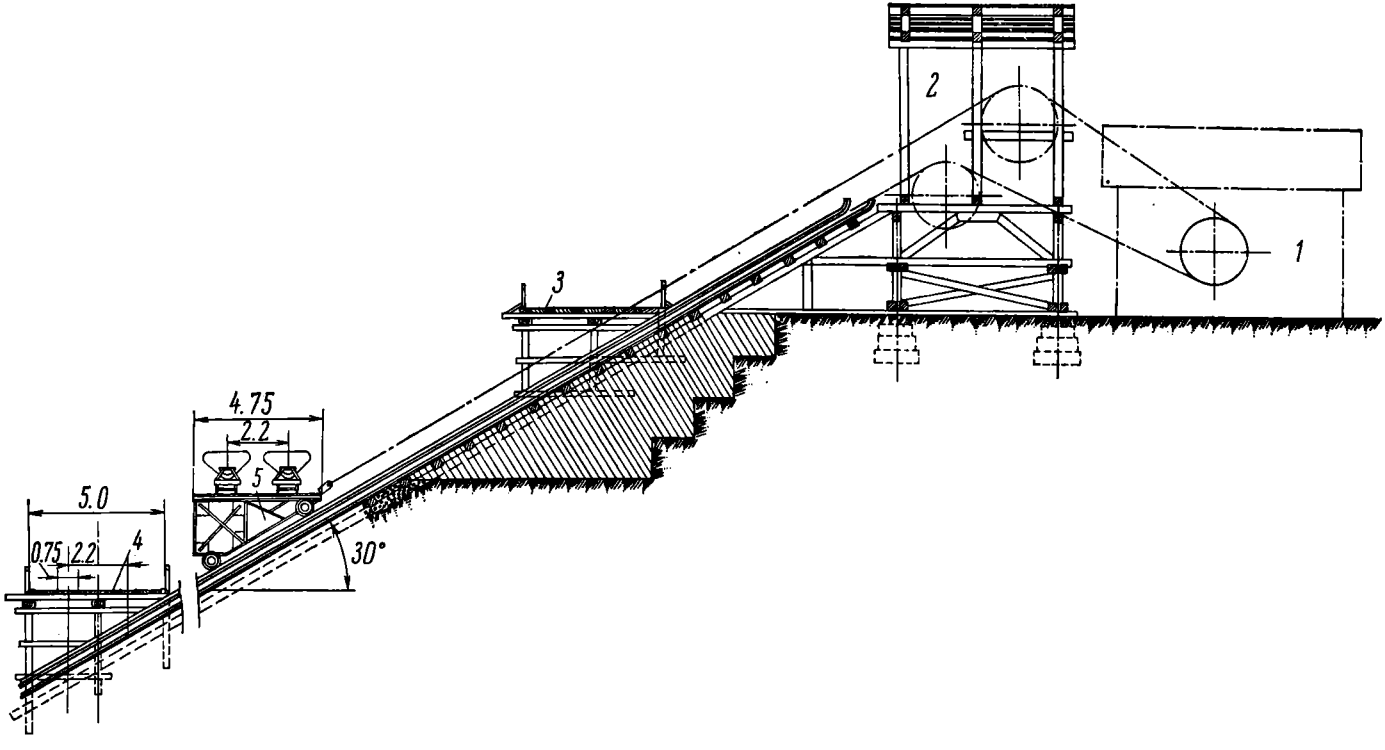
At present tractor-pulled wheeled scraper units with 6-10 cu m buckets are used in the USSR. Scrapers with a capacity of 15 cu m and more are pulled by powerful tractors in conjunction with a pusher. The bucket capacity of wheeled scrapers is now being increased. Scrapers with a bucket capacity of up to 32 cu m and with a power rating of tractors up to 420 h.p. are produced in the USA. Such pulling tractors develop a speed up to 60 km/hr and move the ground over a distance of up to 4 km.

Transportation by rope hoists. Rope hoists may be equipped with skips, cages, endless and tail rope traction, and with pulling cars. Skip hoists are used in working inclined and steep deposits for hoisting the mined bulk from a depth of over 100-150 m. Skips have capacities from 15 to 50 tons. With 50-ton skips and a hoisting height from 100 to 700 m the hoist efficiency reaches 15-5 million tons. Skip hoists usually operate in combination with internal open-pit motor-truck transport.

Cage hoists lift single mine-cars or air-dump trucks from the quarry to the ground surface and lower them back into the pit. The hoisting vehicle is a special cage whose platform carries a mine-car or a motor truck. The angle of the route slope may reach 40-45° with a vertical hoisting height of 500 m. When heavy-duty hoist engines and high-capacity (25-50-ton) cars are used the tonnages handled by cage hoists may be high. They can operate in conjunction with internal open-pit transport. Cage hoists are also employed for hoisting mine-cars of low capacity (1 cu m and less) (Fig. 291). Such hoists handle from 100 to 1,000 tons of load per shift.

Inclined hoists may have one or two carrying platforms, the former type being more popular. In this case a counterweight is used.

The rail track arrangement with a one-way hoist is shown in Fig. 292. Landing platforms resting on piles or on solid rock projections are set up on production horizons of the open-pit to receive mine-cars and roll them into cages. An open-pit may have several


2
1
3
4.75
2.2
5
5.0
0.75
2.2
4
30°

inclined hoists, depending upon the tonnages handled and the arrangement of the hoists. In the Krivoy Rog basin inclined hoists for the delivery of waste are usually installed at the hanging wall, and for the delivery of ore at the foot wall.

Rope hoisting plants with a tail- or endless rope for direct lifting of low-capacity mine-cars (up to 3 tons) are used in small-depth and small-output quarries with a slope angle of not more than 20-22°.

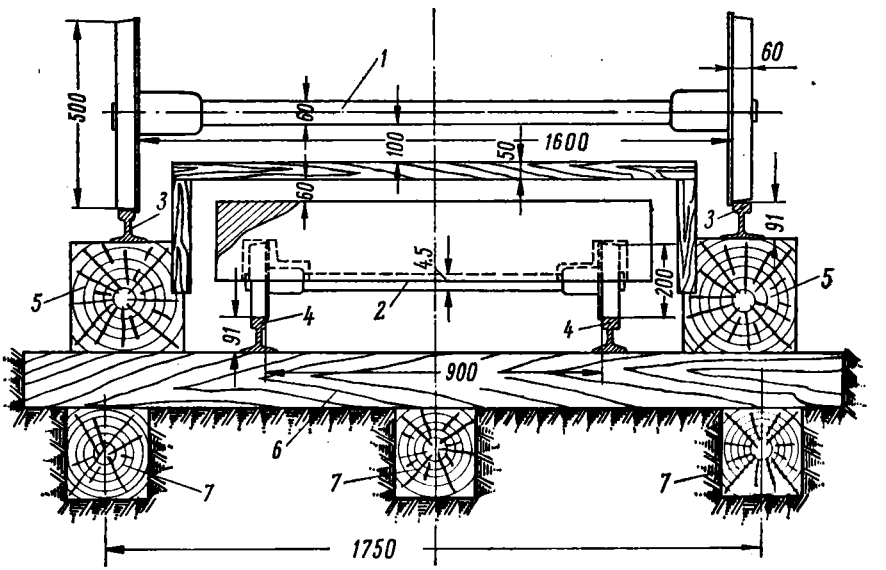


Fig. 292. Rail track arrangement in one-way hoists:
1—wheel axle of platform; *2*—wheel axle of counterweight; *3*—platform guide rails; *4*—counterweight guide rails; *5*—top longitudinal beams; *6*—cross beam; *7*—bottom longitudinal beams

Rope hoists with pulling cars bring up entire trains with coupled cars. In such hoists hauling ropes are fixed not to the head of the train, but to a special tension-carriage, which pushes the train of mine-cars during ascent and backs it during descent. In high-output open-pits use can be made of multirope friction hoists. In this case skips with a load-lifting capacity of up to 70 tons can be used.

Open-pits 100-150 m deep usually employ electric locos and motor trucks. Below a depth of 150 m it is more economical to use combined transport: rail haulage on upper horizons and motor trucks on lower horizons, motor trucks with conveyers, motor trucks with inclined hoists.

Waste dumps and waste disposal. In opencast work large amounts of barren rock are extracted from the earth crust along with the mineral. These rocks are transported to a waste bank and piled up there. Waste disposal is a labour-consuming operation. The rhythmic work of an open-pit largely depends on its timely and proper execution. The most practical way of disposing barren rocks is their dumping into mined-out space. In many cases, however, they have to be hauled away beyond the open-pit boundaries. Waste dumps can be conveniently built by using the local topography—hill slopes, valleys or ravines, by piling rocks on a flat area with the aid of a trestle, a stacking conveyer or an overcasting power shovel. Waste-disposal operations in Soviet open-pits are mechanized.

Waste dumps often assume a segment-, fan- or cone-like (with stacking conveyers) shape. The height of the dump varies from 7-8 up to 60 m, depending upon the physico-mechanical properties of the rock and the mode of stacking. To discharge mine-cars at the dump-site they are spotted at some distance from the high-wall edge. This is achieved by placing the tracks in the corresponding position on the waste bank.

The waste is graded and thrown down over the side of the dump by spreader ploughs. These operate like snow-ploughs and are pulled by electric locomotives. The range of action of the plough mouldboard varies from 1.5 to 3.5 m. The tracks on waste dumps are moved by means of periodic-action track shifters.

At many Soviet open-pits barren rocks are now stacked in waste banks by excavators, and in this case cars and dump trucks are discharged into special receiving trenches near the slope of the dump.

With motor truck haulage the building up of waste dumps is simpler, for the waste in this instance is levelled out by bulldozers.

Permanent waste dumps should be arranged in areas where no mining operations are contemplated.

In estimating the volumetric capacity of waste dumps the initial ground expansion factor is set at 1.10-1.50 and the residual ground expansion factor at 1.05-1.10.

The technical and economic characteristics of waste-disposal operations are listed in Table 37.

Under suitable conditions overburden bridges and belt-conveyer strackers can be successfully employed for waste disposal.

Among the additional methods of waste disposal mention should be made of washing the ground away from the bank slope. In this case the flow-sheet includes a pump plant, a pipeline laid down along the waste bank switchback, branch pipes 100-150 mm in diameter, metal or wooden decking at the point of car or dump truck discharge. The branch pipes are spaced at 1-2 m. The height of the

Table 37

Technical and Economic Characteristics of Waste-Disposal Operations

Mode of dump stacking	Height of dump, m	Efficiency per man shift, cu m	Daily receiving capacity of waste dump, cu m
Bulldozers and motor truck transport	up to 70	500-1,000	1,000-2,000
Spreader plough dumps	10-30	70-150	2,000-3,000
Dumps with power shovels	20-30	120-180	3,000-4,000
Dumps with multibucket excavators	20-40	150-300	5,000-15,000
Dumps with overburden bridges	40-50	250-400	10,000-20,000

receiving bank is set at 7-15 m. Water discharge at a head of 2-3 atm is 1-2 cu m per 1 cu m of ground. Hydraulic disposal of waste is possible only under favourable climatic conditions.

CHAPTER IX

OPENING-UP OF DEPOSITS MINED BY THE OPENCAST METHOD

9-1. GENERAL

By opening-up or early development in opencast mining of deposits is meant the driving of an ingoing trench, which provides access to the deposit from the ground surface, or from an active portion of the deposit to a virgin one, and enables working trenches or cuts to be driven. In some cases a deposit is opened up through underground workings.

The choice of the mode of early development is influenced by: conditions associated with the occurrence of the deposit and its mining, i.e. the shape and size of the open-pit field, the angle of dip, the local topography, the physical and mechanical properties of the enclosing country rocks, the hydrogeological conditions, the type of transport facilities, the disposition of surface structures and plants (concentration plants, warehouses, stockpiles) directly related to mining operations, the system of open-pit mining.

The basic early development excavations in open work are ingoing trenches, whose location and slope angle are the main characteristics of the mode of opening the deposit. The attitude of the ingoing trenches determines the position of working trenches, the working-face front, and the direction of actual mining.

The types of ingoing trenches are indicated in Table 38.

Ingoing trenches should be made with a definite gradient. With electric locomotive traction the maximum permissible slope of trenches is 0.040, and with motor transport 0.150. The maximum permissible slope with conveyer haulage is 18° for belt conveyers, up to 36° for special conveyers, and 25-30° for cars pulled up on rails by a hoist. There are no limitations on the slope angle for cars brought up in cages or skips.

The length of an inclined trench L_t is determined by the depth of its lay H and the slope angle i:

$$L_t = \frac{1{,}000H}{i}, \text{ m}$$

Table 38

Types of Ingoing Trenches

Designation of trenches	Basis for classification	Classification principle
Inside and outside	Location of trenches relative to open-pit contours	Inside or outside open-pit contours
Separate, group or common	Number of benches serviced	One, several or all open-pit benches
Single and twin	Basic purpose	For load and empties; for passage of only loaded or only empty vehicles
Stationary or movable	Permanence of position	Permanent or temporary position
Gently or steeply inclined	Angle of slope	Angle of slope suitable for use of wheel transport
		Angle of slope suitable for use of conveyer haulage or rope hoisting plants

The actual length of a permanent ingoing trench is greater than this because the slope diminishes at curved sections of the road and at sections where the trench adjoins working horizons. Such junctions usually occur on horizontal stretches, the length of the junction section reaching 150-250 m for a standard gauge of 1,524 mm. To estimate the total length of the route the route extension factor can be used. With horizontal junction stretches this factor should be set at 1.40-1.60.

The volume of an inclined permanent ingoing trench with a plane surface is calculated according to the formula:

$$V = \frac{h^2}{i_r}\left(\frac{b}{2} + \frac{h}{3\tan\alpha}\right), \text{ cu m}$$

where h = trench depth, m;
i_r = ruling radient (trench slope), $^0/_{00}$;
b = bottom width of trench, m;
α = slope angle of trench side, deg.

9-2. BASIC METHODS OF OPENING-UP A DEPOSIT

In the working of deposits occurring below the ground-surface level the following methods of early development are practised: through inclined trenches with simply shaped routes of communi-

cation (separate, group or common trenches); through inclined trenches with intricately shaped routes of communication and by making special exit ramps (switchback, loop and spiral ramp roads); through steep trenches (with the use of conveyer haulage or rope hoisting plants); by trenchless stripping; with the use of underground mine-openings; by combined methods.

When mining a deposit occurring above the ground-surface level early development is done through cuts having a trench-and-feather-edge cross-section running along the hillside. The communication routes then have either a simple or complex configuration, depending upon the mining conditions.

Early Development through Inclined Trenches with Simply Shaped Communication Routes

This method is chiefly practised in the working of shallow-seated level and gently dipping deposits. In this case ingoing trenches are usually short and lie outside the open-pit contours, at the flanks

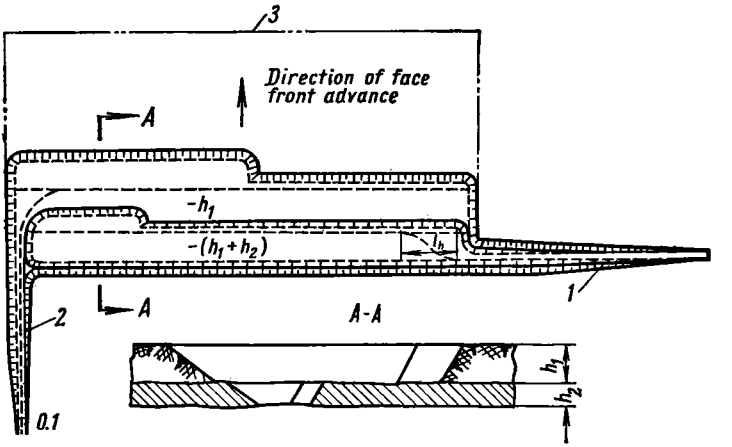


Fig. 293. Opening-up through separate (independent) outside stripping trenshes (plan view):

1—trench opening up production horizons; *2*—trench opening up stripping horizons; *3*—open-pit boundary

of the open-pit field or along the line dividing the field into two wings. Trenches are seldom disposed inside the boundaries of the quarry. The number of benches worked in early development through inclined trenches with simply shaped communication routes usually does not exceed five or six.

Stripping through inclined trenches with simply shaped communication routes is customary in the mining of iron ore, bauxite, phosphorite, limestone, and lignite deposits.

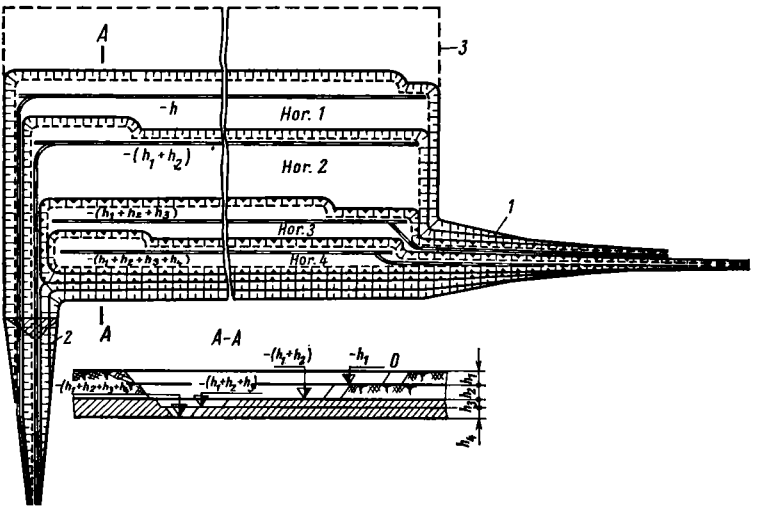


Fig. 294. Opening-up through outside group stripping trenches (plan view): *1*—group of trenches opening up production horizons; *2*—group of trenches opening up spoil horizons; *3*—open-pit boundaries

In opening-up through separate trenches (Fig. 293) the latter are driven from the ground surface down towards the transport sites on the corresponding benches, from which working trenches are started.

The selection of the site for ingoing trenches should be correlated with the permissible road gradients, the minimum amount of stone work and the convenient layout of access roads and waste dumps on the ground surface.

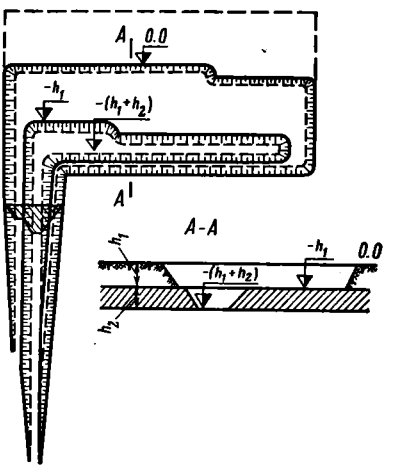


Fig. 295. Opening-up through common (interrelated) outside stripping trenches (plan view)

In opening-up through group trenches (Fig. 294) up to 4-6 active benches may be used. In this case two groups of trenches are usually

driven—one for stripping benches and the other for production benches. As in the first case, the choice of the site for a group of trenches should be in line with the requirement of the highest mining efficiency.

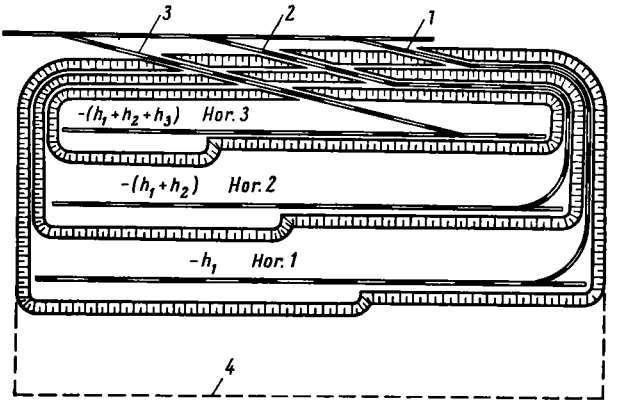


Fig. 296. Opening-up through separate (independent) inside stripping trenches (plan view):
1, *2*, *3*—trenches; *4*—open-pit boundary

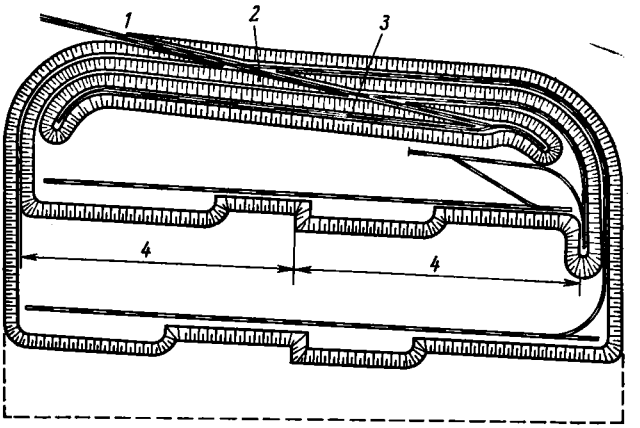


Fig. 297. Opening-up through common (interrelated) inside stripping trenches (plan view):
1, *2*, *3*—trenches; *4*—block

When a deposit is opened up through outside common trenches the concentration of load traffic occurs after the load has been brought to the surface (Fig. 295).

As stated earlier, inclined trenches with a simple layout of communication routes usually lie outside the quarry boundaries. The internal disposition of trenches may be adopted in the working of large deposit, whose dimensions enable a simple route pattern to be arranged within the open-pit boundaries with permissible road gradients. Examples of permanent trenches with a simple layout of routes located inside the open-pit contours are given in Figs. 296 and 297.

Opening-up through Inclined Trenches with a Complex Route Layout and Arrangement of Special Ramps

These methods of early development are practised in the working of thick, gently sloping and steep deposits by deep open-pits with numerous benches (up to 10-12 or more).

Under such conditions of stripping the general network of interdependent trenches lies predominantly within the open-pit boundaries. The total length of roads greatly exceeds the extent of the quarry side, and special ramps have to be provided for the route to pass from one bench to another.

Opening-up through inclined trenches with a complex route layout is usually done in the mining of deep-seated ore and coal deposits.

Early development through inclined trenches with switchback road layout (Fig. 298). The basic conditions for the use of this method are: a steep or inclined dip of the deposit, its considerable thickness and large extent on strike (not less than 600-700 m) with the use of rail transport. The latter condition should be met in order to secure permissible gradients for ramp roads between individual horizons and to enable such ramps to be arranged within the strike of the deposit without driving trenches in stone at the flank of the deposit and without additional excavation of country rock. In this method of opening, tracks are laid on one of the quarry sides, usually at the foot wall of the deposit, with switchbacks arranged on individual horizons. The inclined trench is first driven from the ground surface level E down to the haulage horizon of the first bench. On reaching a pre-set depth (point F) the trench is run horizontally to the deposit flank (point K). The horizontal length FK serves for deploying mining operations and manipulation of trains. After the deployment of work in the first horizon another inclined trench is driven from the horizon dead-end, running in the opposite direction towards the haulage horizon of the second bench, with the tracking continued from the second bench to the third, and so on. A substantial disadvantage of the switchback route layout is the complex organization of train traffic due to the necessity of changing the direction of train travel at each dead-end when trains leave

the open-pit and when they go down from the ground surface; this adversely affects the efficiency of the open-pit, especially with rail transport.

Opening-up through inclined trenches with a loop roadway layout. This method of early development (Fig. 299) is ordinarily employed in the mining of deep-seated deposits with the use of motor transport. An inclined ingoing trench is driven from the ground surface

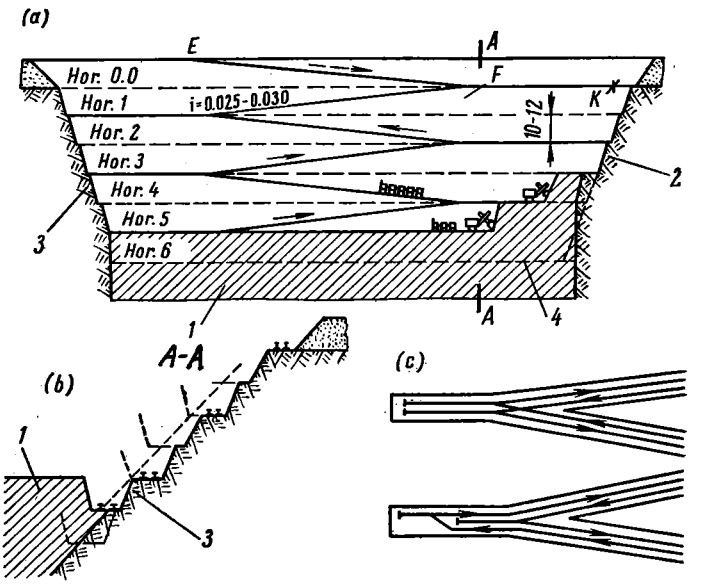


Fig. 298. Opencast operations with a switchback pattern of roadways:
a—section on strike; *b*—section across strike (on *A-A*); c—roadways layout in switchbacks; *1*—mineral; *2*—waste at open-pit flank; *3*—barren rock in foot wall; solid arrow indicates direction of load traffic and dashed arrow, direction of empties traffic

along the non-productive side of the open-pit at a maximum permissible gradient.

At the end of the open-pit field the road line is turned by 180°; this is achieved by arranging a special loop for motor trucks to turn round at a permissible radius of curve. After having carried the trench to the opposite end of the open-pit the route, when necessary, is again turned by 180° (for which purpose a new loop is made), and so on. Figure 299 illustrates opening-up with one loop. To provide the loop space, use is made of projections of solid ground in place; less frequently a special embankment is built. Individual benches are cut up every 10-15 m. Sections of the road located on bench berms are joined to the main road at the corresponding points.

Conditions particularly favouring the use of this method are found in deposits occurring on a hillside, where this method is employed most frequently.

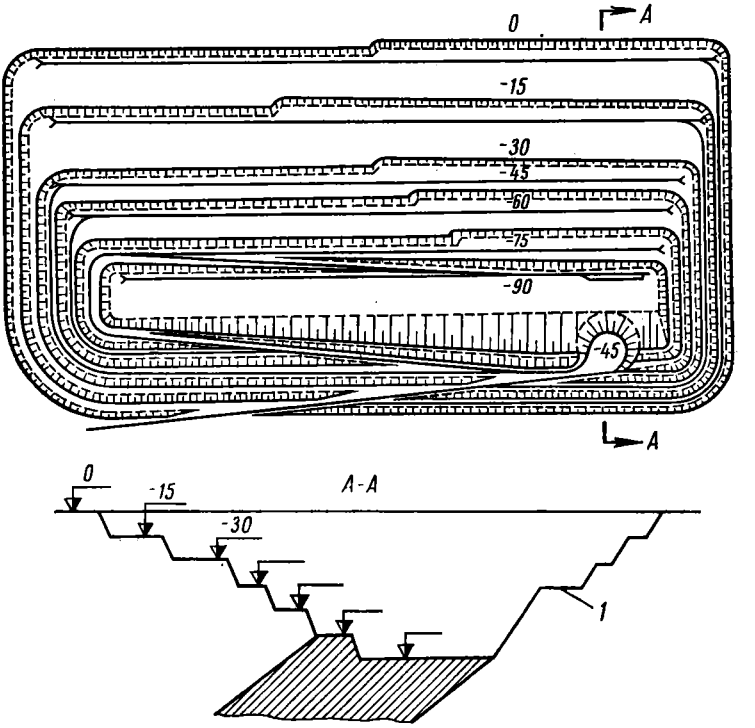


Fig. 299. Opening-up with a loop roadway layout:
1—plot for arrangement of loop-shaped route

Opening-up through inclined trenches with a spiral route layout. The basic conditions for the use of this method are: a level or slightly inclined attitude of the deposit, circular or oval outlines and a considerable size of the deposit.

The second and third conditions stem from the need of providing curves of permissible radius at some sections of the spiral route.

The sequence of driving individual sections of a trench with a spiral route and the deployment of mining operations is as follows (Fig. 300).

The main inclined haulage trench is driven from a convenient spot on the ground surface, selected so as to minimize earthwork in driving and with provision for a possible further development of the roads, the accommodation of ore stock piles and waste dumps.

This trench runs near the contour lines of the projected open-pit down to the first horizon. After the elevation mark of the first bench has been reached a level working trench is pushed forward with the purpose of preparing the horizon for stoping. As the mining in the first horizon progresses the main haulage trench is carried down to the second horizon, this trench being an extension of the one lying above, with a horizontal stretch provided between the sections of the trench. The trench is driven to the second horizon also along the contours of the open-pit, leaving protective berms with the necessary slopes.

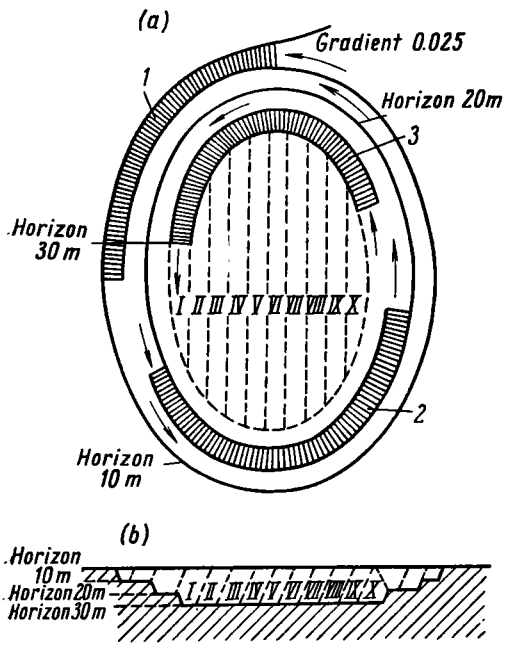


Fig. 300. Opening-up with a spiral route layout:
a—plan view of open-pit; *b*—cross-section; *1*—sloping trench to 10-m horizon; *2*—sloping trench to 20-m horizon; *3*—sloping trench to 30-m horizon

Spiral tracking should be correlated with quarry berms on individual horizons, which usually diminish in size as the pit progresses in depth. One or two track sections of the spiral usually rest upon enclosing country rocks immediately beyond the deposit boundaries.

In practice spiral ramps are frequently used in conjunction with switchbacks—a spiral ramp on the upper benches and switchback ramps on the lower ones.

The spiral arrangement of the route simplifies the organization of train traffic along the banks of the open-pit and ensures a high throughput of the quarry tracks. On the other hand, as the open-pit advances in depth the ingoing trenches run through a new area of the open-pit each time and this involves changes in the direction of mining on production benches. As a consequence the organization of mining operations becomes more complex and the rate of advance in depth reduces. For this reason, the spiral route layout is relatively rarely used in its pure form.

Opening-up through Steep Trenches

This method of early development is employed with conveyer haulage and mechanical hoisting of mined bulk in mine-cars or skips. Preference is generally given to conveyer sets, since these ensure continuous delivery of large amounts of muck in a unit

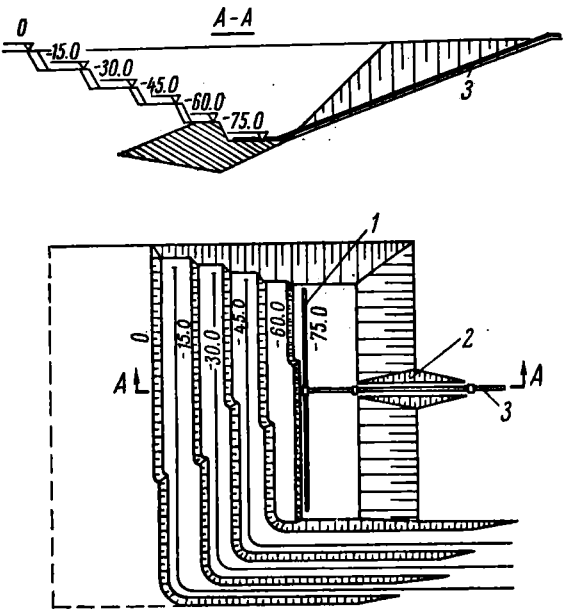


Fig. 301. Early development through a steep stripping trench with a conveyer setup:
1—horizontal belt conveyer; *2*—trench; *3*—inclined belt conveyer

working time. Steep trenches are usually driven along the non-productive side of the open-pit and equipped with several belt-conveyer sets (Fig. 301) (coal open-cuts in the USSR, iron ore open-pits in the USA, and others).

The basic conditions for opening-up through steep trenches with inclined rail hoists are: the limited size of the deposit, multibench extraction, and a great quarry depth.

In these conditions of mining it is either impossible or inadvisable to lay tracks directly from the ground surface into the open-pit because of the large scope of earthwork. Then the mined bulk is delivered from the open-pit by inclined hoists fitted out with special hoisting gear.

Inclined hoists are extended as the mining in the quarry advances in depth. To do this, an inclined ditch is excavated from the lower production horizon up to the elevation mark of the next superjacent horizon. In this ditch a reloading platform is made to which motor trucks or r.r. cars are then pulled up for reloading of waste or ore into skips.

Trenchless Stripping

A distinctive feature of this method is the realization of transport communication between working horizons of the open-pit and the ground surface without driving any trenches. Trenchless stripping, which is rarely used, is practised in the working of deposits by means of cableways, tower cable excavators, etc.

Early Development through Underground Mine-Openings

This method is ordinarily used in mining hillside deposits. Then an adit is driven from the production site and an ore-pass is raised

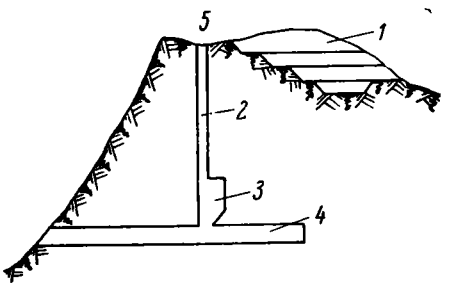


Fig. 302. Opening-up through an adit with an ore-pass:
1—open-pit; *2*—ore-pass; *3*—crusher plant; *4*—adit; *5*—unloading platform

from the adit to the open-pit. A crusher is often set up at the junction of the ore-pass and the adit (Fig. 302).

The ore is brought in motor dump-cars to the unloading platform near the ore-pass. From the latter the ore is passed through the crushing plant, then loaded into r.r. or motor dump-cars and delivered along the adit to the point of destination.

Sometimes an inclined shaft equipped with a conveyer set is driven to open up lower horizons of the open-pit. In the case of a strong and lumpy ore a breaker is installed at the shaft exit facing the open-pit.

If there are underground mine workings within the open-pit area, or if underground mining is contemplated for ultimate exploitation of the deposit, then vertical shafts may be used for early development (Fig. 303).

The expedience of open-pit mining with underground haulage in the working of the bottom horizon of the open-pit (before switching over to underground mining) is proved by the reduced cost of loading the muck into mine-cars (loading by gravity), extraction of the mineral on the bottom horizon of the quarry without additional

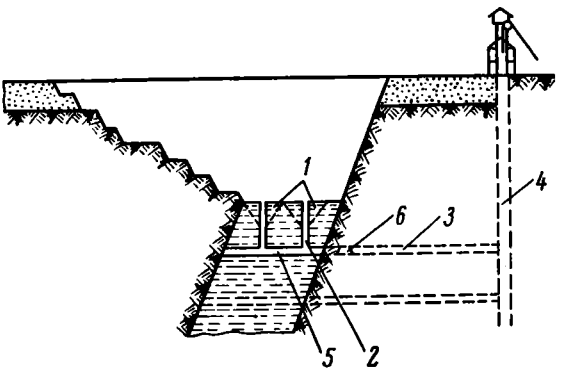


Fig. 303. Opening-up of a bottom horizon in an open-pit by using underground mine workings:
1—glory hole; *2*—ore-chute; *3*—crosscut; *4*—shaft; *5*—crossdrift; *6*—lateral drift

excavation of country rock in the hanging wall, the utilization of the shaft hoist without considerable additional expenditures, and simple organization of work (no haulage tracks or outfall ditches in the open-pit).

Combined Methods of Opening a Deposit

These include two or more basic modes of early development. In mining practice combined methods of opening find wide application, for they suit various conditions of mining individual parts of a deposit.

The following combinations of basic methods of early development are most typical: trenchless stripping of upper stone benches by using heavy-duty excavators with the waste overcast by stackers, and opening-up of production benches through trenches with the use of railway or motor truck haulage; stripping of stone benches through inclined trenches and early development of production benches by driving steep trenches (with the mineral delivered by conveyers); a combination of different inclined trenches, viz. separate, group and common trenches; a combination of trenches with switchback and spiral route layouts; combined opening-up of the top portion of the deposit throughtrenches and of the lower portion

through underground workings; a combination involving early development of upper benches through trenches and trenchless opening-up of lower benches with loading by excavators below the ground level. Other combined methods of stripping may also be used.

Stripping of Deposits Occurring Above the Elevation Mark of the Adjoining Ground Surface

It has been mentioned earlier that such deposits are opened up either through underground workings or through box cuts of a trench and feather edge cross-section driven on a hillside with a simple or complex route layout, depending upon the mining conditions, the

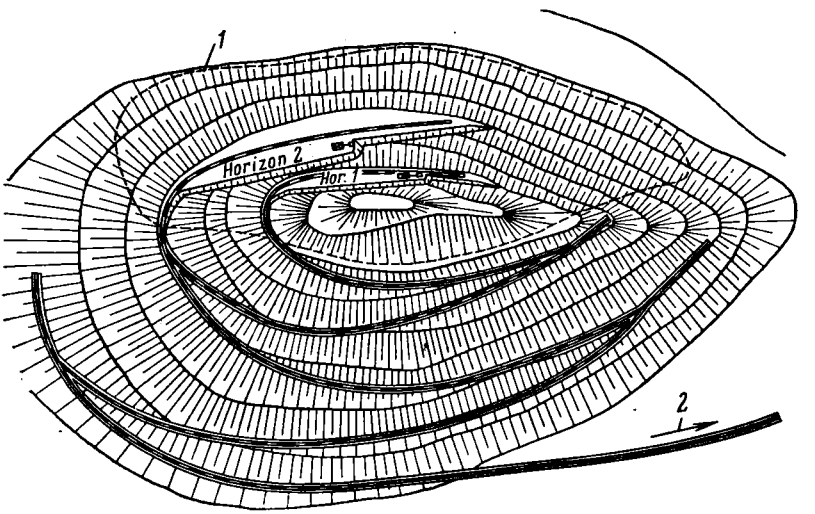


Fig. 304. Layout of a switchback route on a hillside:
1—ore body boundary; *2*—to concentration plant

position of the working horizons in the deposit, the lay of land, and the type of transport facilities employed at the permissible limiting gradients of haulage roads.

The route of permanent box cuts usually runs outside the bounds of the mineral deposit and has an intricate shape (loop-like, switchback, less frequently spiral or combined). Stripping through inclined box cuts with a switchback route layout is illustrated in Fig. 304.

In open work on a hillside each stone bench or a group of stone benches can be serviced by local outside waste dumps sited directly on the hillside (Fig. 305).

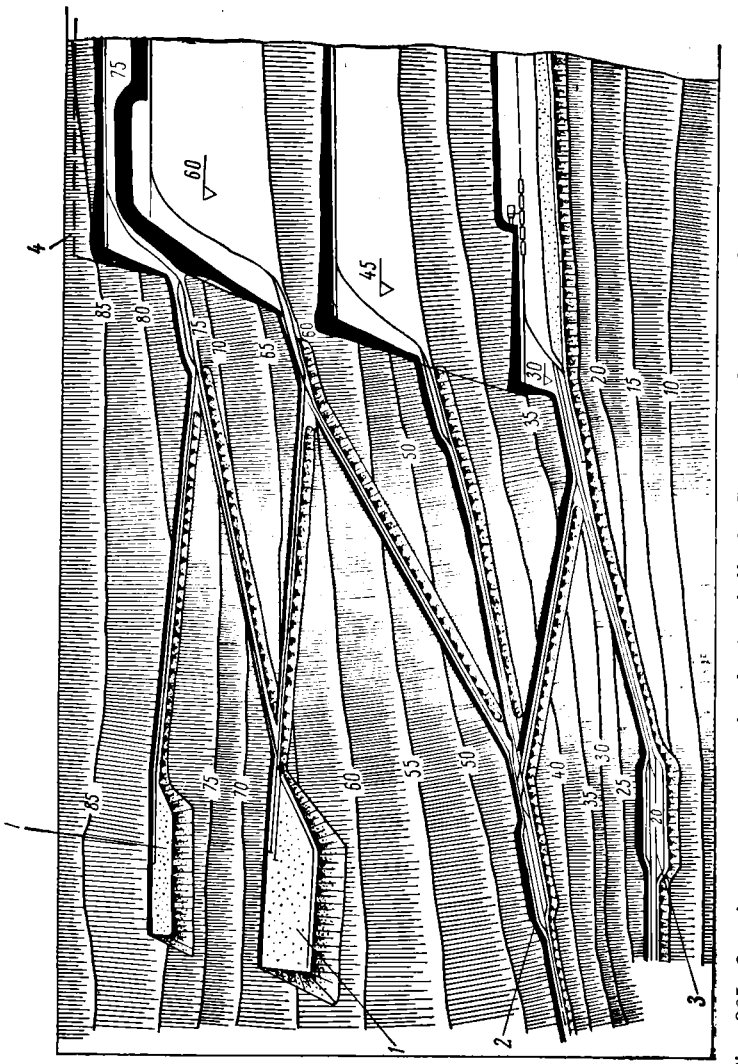


Fig. 305. Opening-up on a gently sloping hillside. Permanent box cuts of a trench and feather edge section with external switchback layout:

1—local waste dump; *2*—by-pass siding; *3*—open-pit terminal station; *4*—open-pit field boundary

An example of mining a hillside deposit with drivage of a system of entrance box cuts is the working of the Magnitogorsk iron ore deposit.

9-3. FINAL CHOICE OF THE MODE OF STRIPPING

The selection of one of the basic or combined modes of opening-up a deposit should be correlated with the choice of the transport facilities and the ultimate decision should be based on a technical and economic comparison of rational variants with due consideration for initial outlays and operating expenses, dates fixed for the commencement of exploitation and the eventual production capacity of the open-pit. The contemplated method of mining should also be taken into account.

In most cases questions associated with opening-up and with methods of mining a deposit are interrelated and are resolved jointly.

CHAPTER X

MINING SYSTEMS AND MINING OPERATIONS IN OPEN-PITS

10-1. GENERAL

By a system of mining in open work are meant safe and economical methods of implementing a definite set of stripping, stoping and other mining operations ensuring the attainment of the target output of an open-pit with rational utilization of the deposit reserves.

Academician N. Melnikov and Prof. E. Sheshko, Dr. Sc. (Eng.), base their classification of the mining systems used in the working of deposits by opencast methods on the method of conducting the most labour-consuming stripping operations. The latter, in their turn, are characterized by the mode of waste disposal and the location of waste dumps (see Table 39).

Table 39

Classification of Systems of Opencast Mining

Groups of mining systems	Description of mining system	Designation of mining systems (Fig. 306)
A. Systems involving overcasting of overburden [1] (without haulage) and course stacking	1. Systems with direct side-casting of overburden	A-1
	2. Ditto, with multiple overcasting of overburden	A-2
	3. Ditto, with overcasting by means of conveyer stackers or overburden bridges	A-3
B. Systems involving haulage of overburden and its disposal along waste dumps	4. Systems with haulage of waste to inside dumps	B-4
	5. Ditto, to outside dumps	B-5
	6. Ditto, to inside and outside dumps	B-6

[1] By overburden are meant barren country rocks which are subject to stripping. Stripping is removal of waste to gain access to the useful mineral.

Groups of mining systems	Description of mining system	Designation of mining systems (Fig. 306)
C. Combined systems of mining with overcasting and haulage of overburden. With disposal along and across waste dumps	7. Systems with overcasting of waste to inside dumps and partial haulage to outside dumps	C-7
	8. Systems with haulage of waste to outside dumps and partial overcasting to inside dumps	C-8
D. Mining systems involving insignificant amount of overburden stripping, where methods of waste disposal are of no essential importance		A-0

This classification differs from that accepted for underground mining in that it does not fully reflect the sequence of development operations and of actual extraction of the mineral. This drawback is eliminated in a classification devised by V. Rzhevsky, Corresponding Member of the Academy of Sciences of the USSR, who has suggested that, apart from the location of waste dumps, the classification of systems should take into account the direction of bench extraction in plan (longitudinal, lateral, fan- and ring-shaped) and the direction of actual mining in cross-section (in horizontal, inclined or steep slices). The enumerated factors are details (groups, subgroups), which should be considered in a more comprehensive characteristic of each of the systems listed below (these are reviewed in a definite sequence, according to the main feature—the methods of overburden stripping and waste disposal).

10-2. METHODS OF MINING

A. SYSTEMS INVOLVING OVERCASTING OF OVERBURDEN WITHOUT ITS HAULAGE

1. METHODS WITH COURSE STACKING AND MULTIPLE OVERCASTING (RECASTING) OF OVERBURDEN (A-1, A-2)

This system (Fig. 307) is used in mining shallow-seated thin and medium-thick horizontal bedded deposits (bauxites, phosphorite ores, coal, refractory clays, etc.).

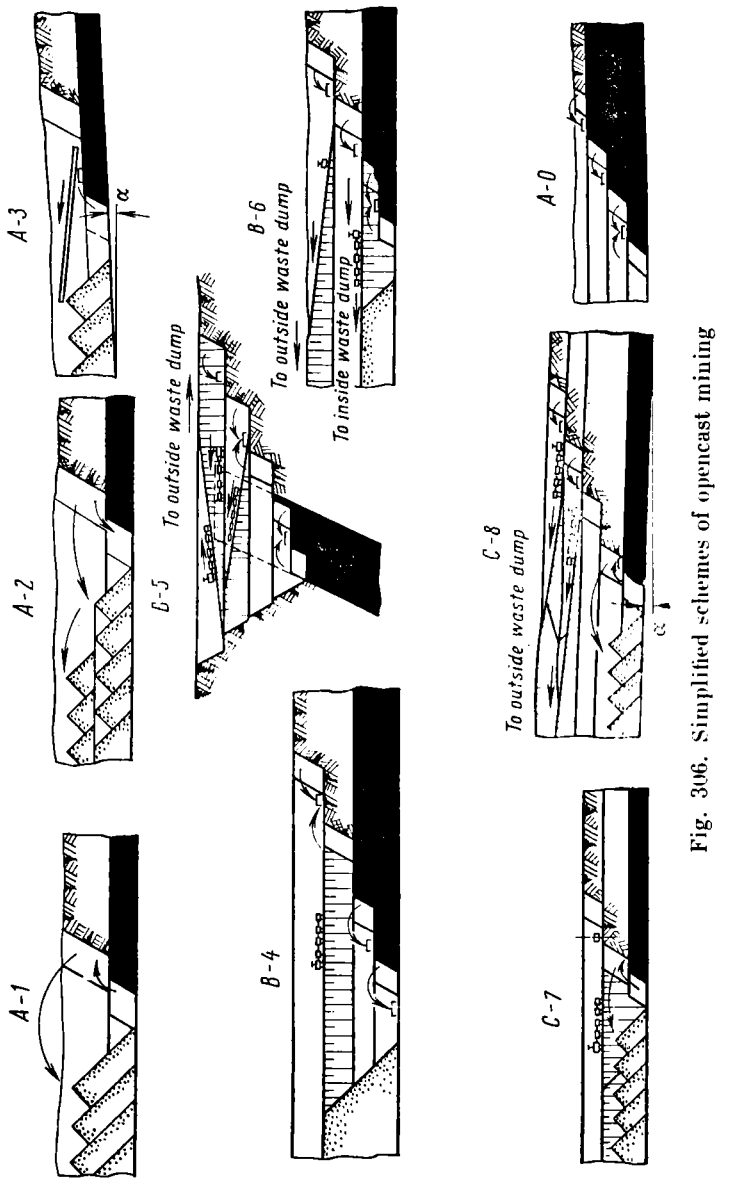


Fig. 306. Simplified schemes of opencast mining

The thickness of cover rocks should not exceed the value permitting the operation of a power shovel or a dragline (insofar as their linear dimensions are concerned). For power shovels of ordi-

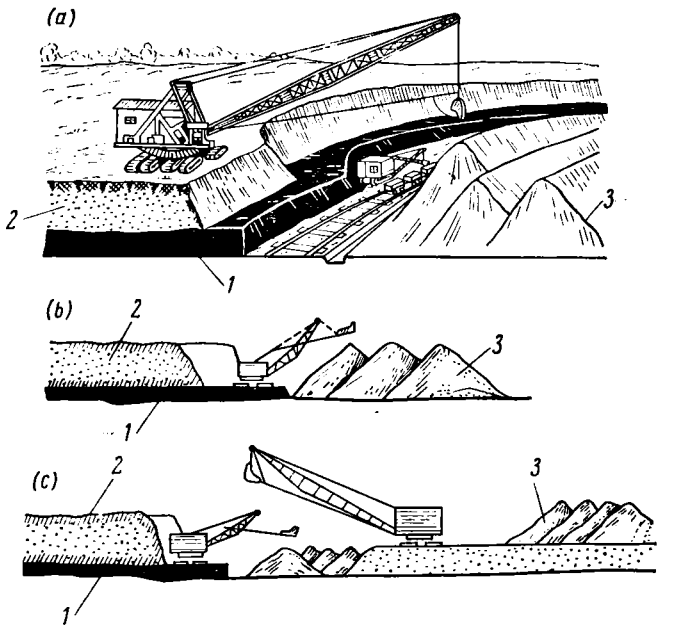


Fig. 307. Systems of overcasting of stripped ground by excavators:
a—overcasting by stripping draglines; *b*—overcasting by power shovels; *c*—recasting of stripped ground; *1*—mineral; *2*—barren rock; *3*—waste banks

nary size it should not be more than 5-10 m and for big excavators, 18-22 m. With course stacking of waste the stripping ratio may be 12 m^3/ton and more.

To extract the mineral, a special production-type face power shovel is used.

In the system with multiple overcasting of the stripped waste two excavators are in operation: one does the stripping (power shovel) and the other (dragline) is engaged in overcasting the excavated ground. The method is applicable with stable ground, not susceptible to slipping. One of the excavators is mounted on the primary waste dump site.

In selecting the system with multiple overcasting it is necessary to consider that in working soft grounds the cost of hauling the overburden to outside dumps is approximately the same as in recasting.

2. SYSTEM WITH OVERCASTING OF STRIPPED GROUND BY CONVEYER-STACKERS (A-3)

Here (Fig. 308) two independent stripping operations are performed: excavation and stacking.

The waste, which is commonly taken out by multibucket excavators, is delivered to inside dumps by belt conveyers mounted on a

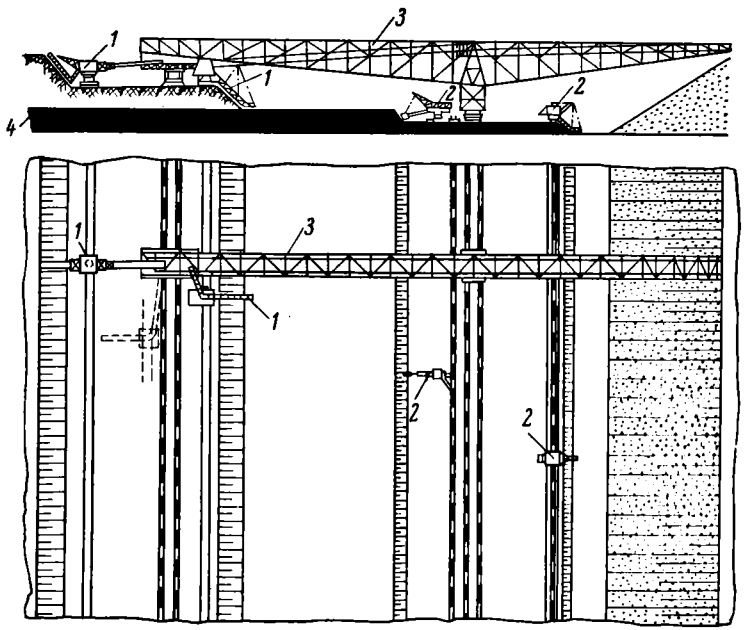


Fig. 308. Overcasting stripped ground by a conveyer-stacker:
1—excavators stripping overburden; *2*—excavators digging mineral; *3*—conveyer-stacker; *4*—mineral

special discharge boom (cantilever) or on a special truss (in overburden bridges).

The waste is moved from the working face to the waste bank over a distance of 50-100 m with a conveyer-stacker and over 300-500 m with an overburden bridge.

The main conditions for the use of systems involving overcasting of overburden by stackers are: horizontal occurrence of the mineral bed, regular hypsometry of the bed roof, soft and dry cover rocks, considerable reserves of the mineral.

Overburden bridges are capable of handling from 400 to 2,000 cu m of ground per hour, depending upon the width and velocity of the belt. The overburden bridge consists of the main and cantilever bridge trusses mounted on two supports with a wheel or caterpillar carrier arrangement. The main and cantilever trusses carry belt conveyers. The ground is taken out by multibucket excavators and then transferred onto conveyers. When in operation, the excavators and the overburden bridge move along the benches. As these are worked out the tracks and the whole structure are shifted to a new site. The annual output of open-pits equipped with overburden bridges runs into tens of millions of cubic metres of muck. Several high-capacity open-pits equipped with overburden bridges are in operation in the USSR.

B. TRANSPORT SYSTEMS OF MINING WITH HAULAGE OF OVERBURDEN

3. MINING METHOD INVOLVING HAULAGE OF OVERBURDEN TO INSIDE WASTE DUMPS (B-4)

This system (Fig. 306) finds application in the working of flat, deep-seated deposits of medium- and above-medium thickness with open-pit fields of considerable extent along the strike.

Like in other systems with inside waste dumps the mandatory condition here is the mining of the deposit throughout its thickness right from the start. The waste is hauled by railway and motor vehicles from the working faces to the waste banks. The elevation marks of stripping and dumping sites should be so correlated that the waste goes from the faces to the dumps down the grade or at least in a horizontal direction.

4. METHOD OF MINING WITH WASTE DISPOSAL IN OUTSIDE DUMPS (B-5)

This system (Fig. 306) is usually employed in mining steep and inclined deposits or very thick flat deposits. Here mining proceeds from the foot wall to the hanging wall and progresses in depth gradually. Inside dumps cannot be made here, since the floor of the open-pit does not become exposed until the last period of quarry operation. This method is widely used in the USSR. It may be applied in most diverse conditions of mining.

C. COMBINED SYSTEMS OF MINING WITH OVERCASTING AND HAULAGE OF OVERBURDEN

5. SYSTEMS WITH OVERCASTING (HAULAGE) OF WASTE TO INSIDE (OUTSIDE) DUMPS AND PARTIAL HAULAGE (OVERCASTING) OF WASTE TO OUTSIDE (INSIDE) DUMPS (C-7, C-8)

Combined methods (Fig. 306) represent combinations of basic systems. They are used under certain conditions of mining when application of any single system is either impossible or inadvisable. For example, combined mining and partial overcasting of waste to inside dumps is practised when it is difficult to establish communication lines for deep-lying benches. A combined system then yields better technical and economic results by comparison with an ordinary method of mining with haulage of waste to outside dumps.

6. FIGURES ILLUSTRATING ANNUAL ADVANCE OF WORKING FACES AND MARGINAL STRIPPING RATIOS FOR DIFFERENT METHODS OF MINING

The intensity of bulk extraction is related in a definite way to the mining method. It should be borne in mind that in order to achieve uniform mineral output in an open-pit the front of working faces

Table 40

Mean Annual Working-Face Advance and Mean Marginal Stripping Ratio in Mining Coal Deposits
(According to E. Sheshko)

Mining system designation (Fig. 306)	Annual advance of working-face front, m	Marginal stripping ratio, m^3/ton	Note
A-1	250-400	10-12	For seams up to 2.0-2.5 m thick
A-2	—	5-12	Depends upon the number of recastings
A-3	200-250	6-10	—
B-4	150-200	5-7	With electrified railway transport
B-5	100-150	3-5	
B-6	—	4-5	
C-7	200-300	5-8	With top cover rocks removed by using trackless transport
C-8	100-150	4-6	With electrified railway transport

should be different for different systems of mining. The minimum front is associated with methods involving overcasting of waste into worked-out space, the maximum front—with those involving outside waste dumps. Highly efficient excavation of the mined bulk also permits of high stripping ratios (Table 40). In the mining of coal deposits the mean annual advance of the working-face front varies from 100-150 m (breakage, excavation, out-of-mine transport) to 250-400 m (breakage and excavation with overcasting). The marginal stripping ratio ranges from 3-5 to 10-12 m^3/ton, respectively.

In the working of ore deposits with excavation of rocky ground the marginal stripping ratio and the annual advance of the working-face front are smaller. The mean annual advance of the working-face front in the horizontal direction may be tentatively set at 50-100 m. The marginal stripping ratio usually varies from 5-6 to 8-10 m^3/m^3, reaching even higher figures in some instances. A possible further rise in the marginal stripping ratio should be expected.

10-3. OPENCAST MINING

Driving of trenches. The method of driving trenches is largely determined by the physical properties of the ground, its cross-section and the equipment used.

Trenches can be dug by excavators (with and without the use of transport facilities), by wheel scrapers, by using hydromechanical devices or by bulk pin-point blasting.

Cutting of trenches without haulage of the excavated rock (course stacking) (Fig. 309) is practised in soft ground by using a power shovel, when the whole of the excavated ground can be disposed of on the trench edges, or subsequently recast by the shovel over a short distance. This method enables a trench to be pushed forward in a short time and at a low cost. With course stacking a trench can be excavated simultaneously at several places along its length. The time utilization factor of the power shovel may then reach 0.9-0.95, instead of 0.35-0.40 in the case of hauling the waste over considerable distances with complicated shunting and switching operations. This method of trench digging, however, is possible only in stable ground that would rule out the caving of the trench edges.

In digging trenches with the haulage of the excavated ground the following types of transport are used: railway vehicles (when large amounts of excavated material are hauled over long distances), and motor trucks (when the amounts of material are small and the haul distances moderate).

In this case a trench may be driven simultaneously over its full face (with a switchback layout), by splittings in a single strip or by

splittings in several strips. In the last two methods the face advances consecutively in separate areas of the trench cross-section (Fig. 310).

Full-face advance of trenches is practised when their width and depth are small and it is expedient to do the job in a single face (Fig. 311). With this method transport vehicles are moved to the

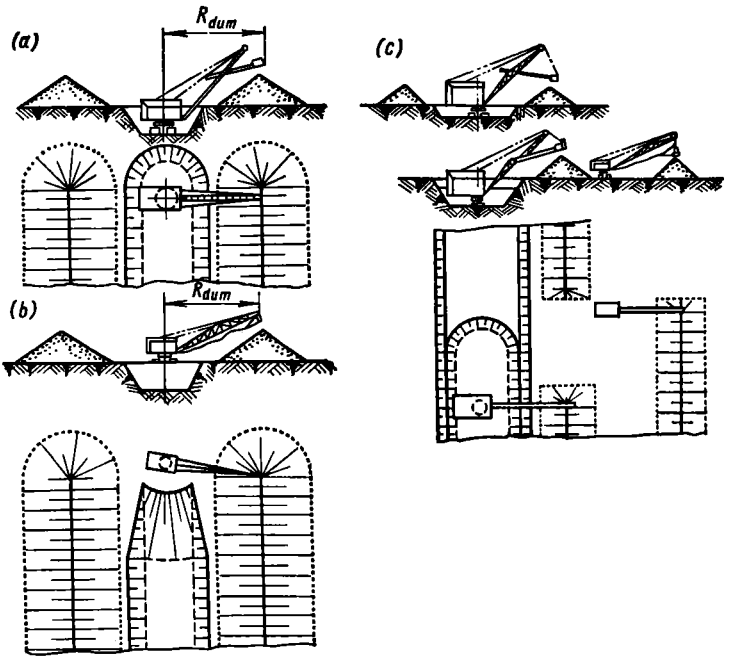


Fig. 309. Driving trenches by course stacking:
a—by power shovel installed in trench; *b*—by dragline; *c*—by power shovel with subsequent recasting of waste by dragline

working face into the dead-end over tracks laid on the floor of the opening (when a power shovel is used), or on the edge of the trench (when work is done by a dragline or a power shovel with an extended boom).

Driving by splittings in a single strip is practised in wide trenches, when full-face digging presents technical difficulties. The trench is then split into several parts from one side to the other and these are excavated successively along the entire length of the trench or over a considerable part of it.

In making splittings the tracks may be run throughout the whole length of the trench and power shovels with long booms used, which

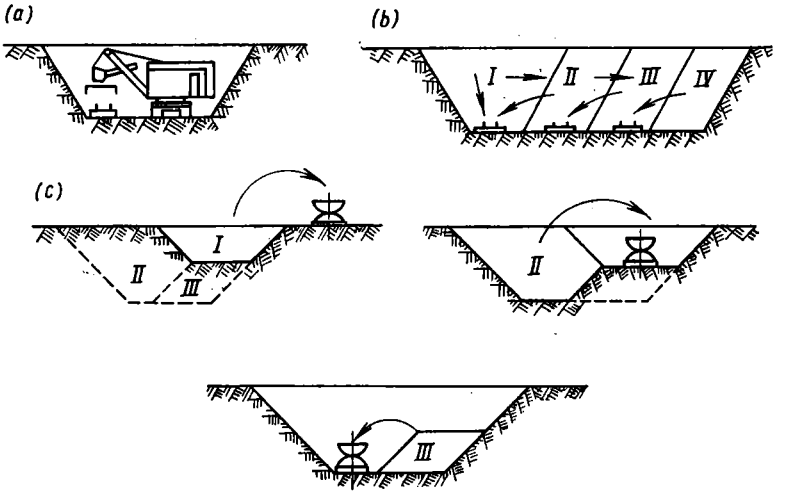


Fig. 310. Transportation schemes for trench digging:
a—full-face drivage; *b*—drivage by splittings in a single strip; *c*—drivage by splittings in two strips (*I*, *II*, *III*—sequence of trench digging)

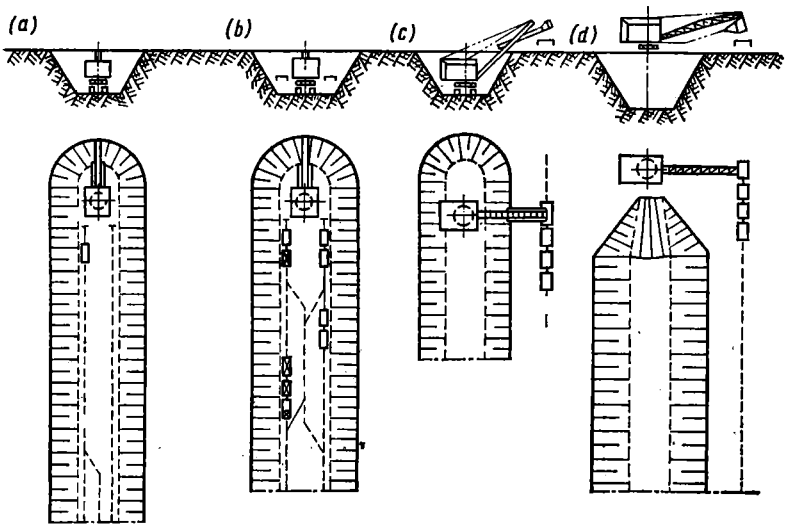


Fig. 311. Full-face drivage of trenches by power shovels with the use of rail transport facilities:
a and *b*—with rail track laid in trench; *c* and *d*—with rail track laid on trench edge

simplifies the delivery of empties to the face and ensures efficient digging.

Drivage by splittings in two (more seldom three or four) horizontal slices is practised when the depth and width of the trench are considerable, and then the work is done with power shovels (Fig. 312).

In rocky ground slice-wise drivage of trenches is successfully done by using multirow millisecond-delay blasting. A large section of

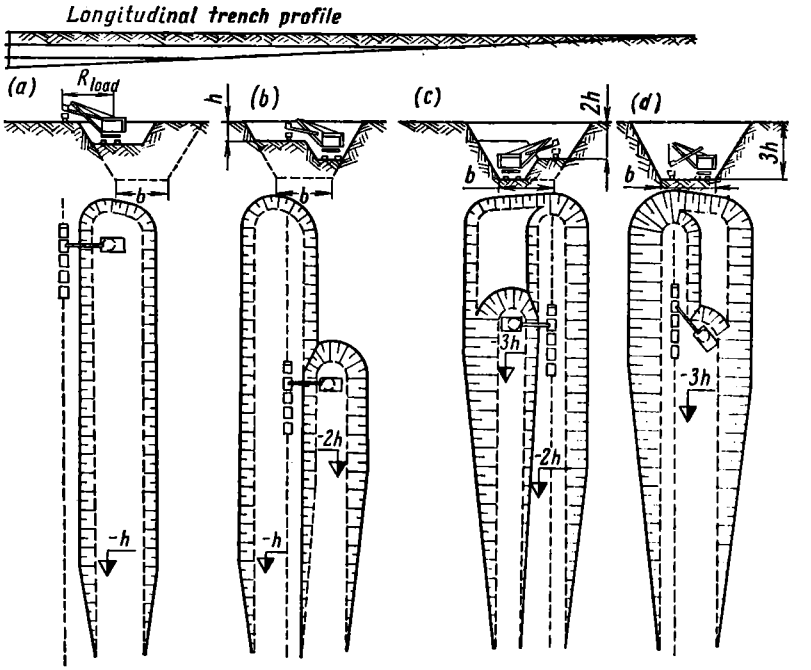


Fig. 312. Driftage of a sloping trench by slicing:
a—excavation of the 1st slice; *b*—excavation of the 2nd slice; *c*—excavation of the 3d slice; *d*—additional excavation of the 3d slice; *b*—bottom width of trench; *3h*—total depth of trench; *h*—height of one slice; *2h*—height of two slices

the trench (200-400 m) is then blasted out to its ultimate depth. The loosened rock is excavated in slices with bank face-to-face loading into r.r. or motor dumpcars. This greatly speeds up the driftage of the trench.

A new slice-wise method of cutting an ingoing trench in a waterlogged ground has been used at the open-pit of the Ingulets Mining Combine. After a bulk multirow millisecond-delay blast in a section 200 m long the trench was carried on in slices from the underlying to the upper horizon. In this way the rate of advance of the trench was increased more than twofold.

The driving of trenches by the combined method (course stacking for the upper benches and transportation of the waste for the lower benches) is done when the scope of work is large, the trenches are deep and the earth-moving distances considerable.

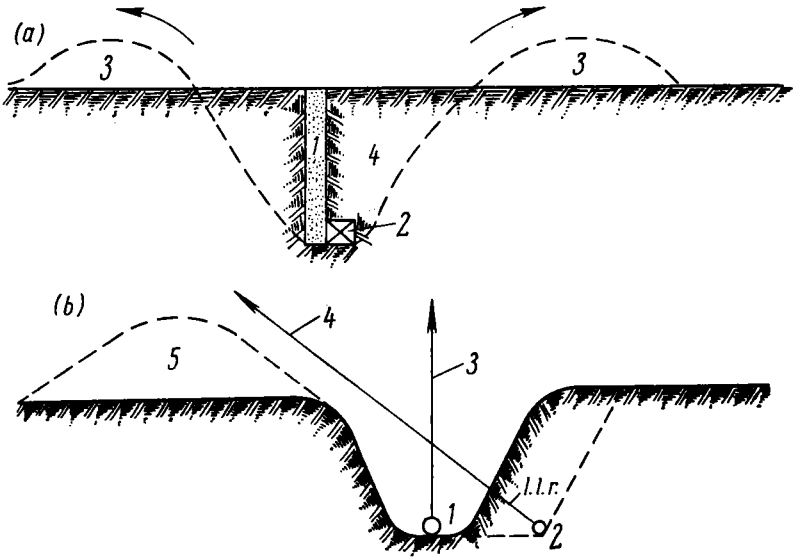


Fig. 313. Drivage of a trench:

a—with two-side pin-point blasting: *1*—earth-filled pit; *2*—tunnel charge for coyote blasting; *3*—ground thrown away after pin-point blasting; *4*—projected trench cross-section; *b*—with oriented blasting (section before second blast): *1*—first-blast charge; *2*—charge to be fired after first blast; *3*—direction in which ground is thrown aside by first blast; *4*—direction in which ground is thrown aside by second blast; *5*—waste bank position after second blast

The cutting of trenches by tractor scrapers is done in soft grounds with small amounts of the material handled and short hauls of the waste.

Hydromechanical drivage of trenches is used in soft grounds where water, cheap sources of electric power and suitable sites for hydraulic refuse dumps are available.

Pin-point blasting of trenches (Fig. 313) is a highly efficient method of trench drivage and is of particular importance in developing regions where no sources of electric power and needed transport facilities are available in the initial period of the opencast work.

In this method bore-pits with tunnels containing large amounts of explosives (to be fired simultaneously) are sunk all along the route of the projected trench. The blast throws the rock away on both sides over a considerable distance from the trench. If the ground has to be thrown aside in one direction, so-called oriented blasting is

done. In this case two charges are placed, the explosives being fired in close succession so that the rock shot up by the first blast is caught up in the air by the force of the second blast and is thrown aside in a pre-determined direction together with the rock lifted by the second blast. The direction of the throw is determined by the orientation

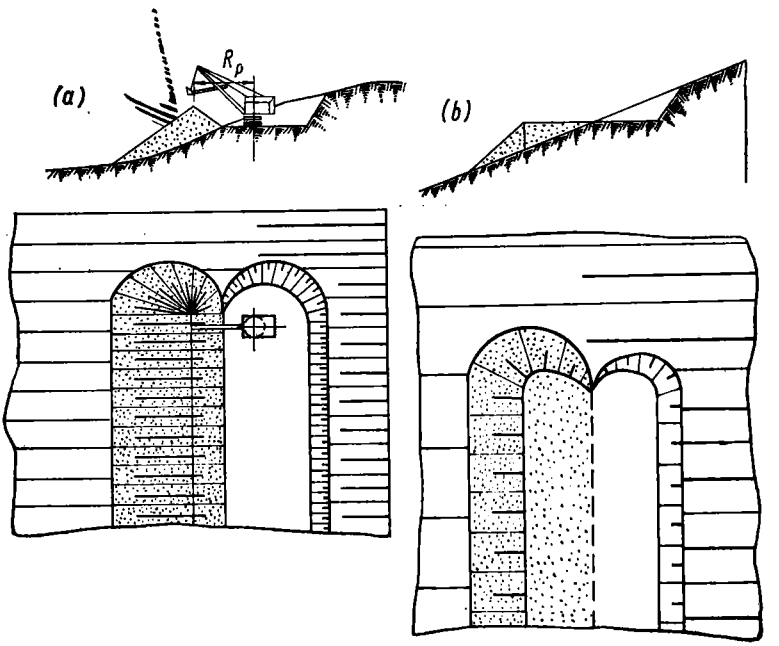


Fig. 314. Driving a trench on a hillside:
a—driftage diagram; *b*—transportation berm on gently sloping hillside

of the line of least resistance of the second charge, which acts after an exposed surface has been formed by the first shot.

Pin-point blasting is widely practised in the USSR in driving trenches and railway cuts.

In driving a working trench 1,000 m long at the Korkino open-pit with a top width of 80 m and a depth of 20 m some 800 thous. cu m of ground was shot up at a time. The explosives consumption amounted to 2.25 kg/m^3 (with the ground consisting of moist clay).

Cuts of a trench and feather-edge cross-section on a hillside are usually made by power shovels, the muck being disposed of on the versant. The excavated ground can be utilized for building a transportation berm (Fig. 314).

The basic technical and economic characteristics of trench driving are listed in Tables 41 and 42.

Table 41

Minimum Bottom Width of Ingoing Trenches

Types of transportation	Trench width, m	
	One-way	Two-way
Railway, track gauge 1,524 mm	7.6-8.0	11.7-12.1
Motor road (depending upon the overall dimensions of the truck)	4.5-6.8	7-14
Conveyer sets (depending upon the belt width)	2.5-3.0	4.0-4.5

Note: The width of a trench should be in line with the parameters of the drifting machinery and the drivage conditions (when a power shovel is used the bottom width of the trench should be at least twice the slew radius of the shovel body so as to provide optimal loading conditions. For the ЭКГ-4 power shovel it should be 22 m).

The rate of advance of a trench. The average rate of advance of a trench varies from 80 to 120 m/month. It may be increased appreciably by using multirow millisecond-delay blasting on the cut-hole pattern and with adequate organization of excavation operations (Fig. 315). A good example of trench driftage by pin-point blasting is the cutting of a trench at one of the Krivoy Rog basin open-pits 525 m long in the space of 2 months and 25 days with the average power shovel efficiency of 35 thous. cu m per month. Only 10 blasts of 25-30 blastholes each were made during the drivage

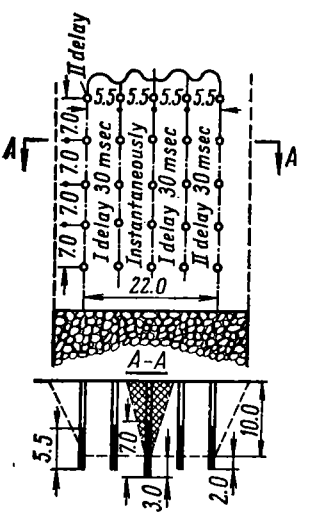


Fig. 315. Multirow millisecond-delay blasting with the muck left at the face in driving a trench in one of the Krivoy Rog basin open-pits

of this trench. At another open-pit a trench 200 m long was driven in 22 working days.

Table 42

Gradients of Ingoing Trenches

Type of transportation	Prevalent gradients
Inclined trenches	
Railway (electromotive traction)	0.030-0.040
Motor roads	0.060-0.100
Steep trenches	
Pusher hoist without cages	0.125-0.250
Belt conveyer	0.250-0.330
Skip hoist	0.500-1.000

10-4. DEWATERING OF OPEN-PITS

Water inflow in an open-pit may be due to surface and ground waters contained in the solid ground and rocks *in situ*. To preclude flooding of the open-pit and to ensure normal and safe conditions for the conduct of operations the open-pit must be protected from surface waters, and the solid host rocks and the mineral should be properly drained. Dewatering is of tremendous importance for the stability of bench high-walls and open-pit banks. To protect an open-pit from surface waters hillside ditches are dug on the ground surface and drain ditches inside the open-pit. The gradient of such ditches is set at 2-3‰. Hillside ditches are made on the side of natural runoff. To collect precipitations finding their way into the open-pit and also to catch water coming from other sources, drain ditches are cut in the open-pit floor, along which the water is directed to headers and then pumped up to the ground surface (Fig. 316).

The maximum inflow of precipitation water in an open-pit can be estimated by the formula:

$$Q = C \cdot a \cdot F\psi, \ \text{m}^3/\text{sec}$$

where C = climatic coefficient characterizing estimated rainwater inflow in given area, $\text{m}^3/\text{sec} \cdot \text{km}^2$;

a = coefficient taking into account runoff delay and absorption of water by soil (a = 0.017-1.25, depending upon extent and slope angle of catchment area);

F = catchment area, sq km;

ψ = coefficient taking into account different water permeability of soils (ψ = 0.5-1.3).

Preliminary dewatering of solid rocks and the mineral in place can be effected by the surface, underground and combined methods. In the surface method drainage is realized by making through cuts

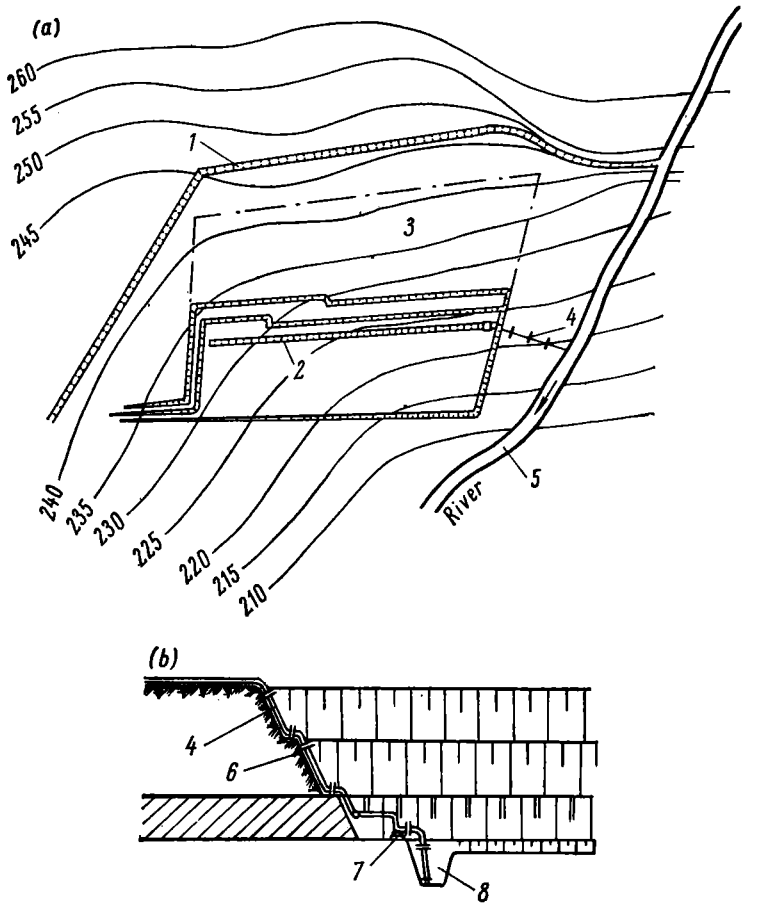


Fig. 316. Layout of ditches and pumping plant:
a—hillside and drainage ditches; *b*—pumping plant; *1*—hillside ditch; *2*—drainage ditch; *3*—open-pit field; *4*—pipeline; *5*—river; *6*—open-pit bank; *7*—pump; *8*—water-collecting header

(in pits of small depth), drilling horizontal drain boreholes on the quarry banks, or setting up suction filters. In the filter variant vertical tube-encased boreholes are sunk from the ground surface down to aquifers, the water being pumped out by special deep-well pumps with a discharge capacity of 30 to 400 m^3/hr, a head of up to

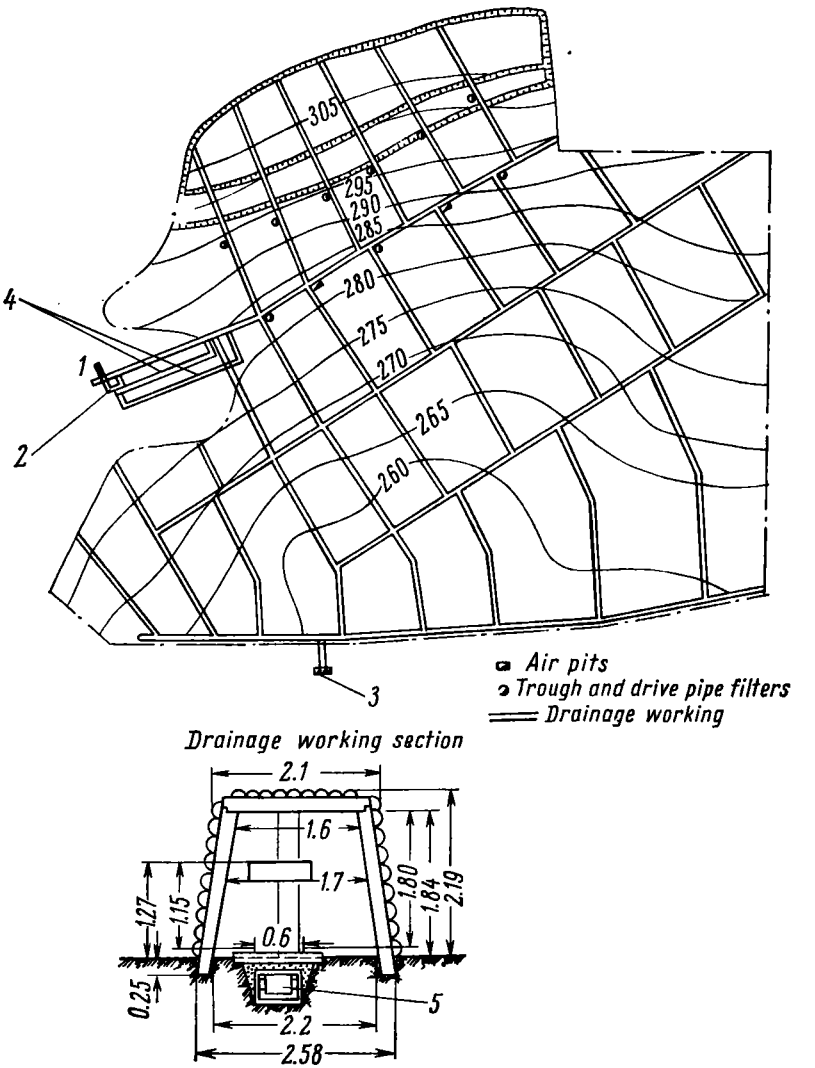


Fig. 317. Layout of drainage workings in the underground method of dewatering a deposit (plan view):

1—shaft No. 1; *2*—pump room; *3*—shaft No. 2; *4*—main sump; *5*—outfall ditch

80-120 m and a pressure pipeline 126-305 mm in diameter. Suction filters are used in dewatering sandy and similar grounds readily giving off water. The filters are spaced at 50-150 m.

The underground method of dewatering is employed in extremely water-logged deposits. A network of underground openings are driven every 200-250 m along the quarry banks over the deposit or beneath it. These drain tunnels communicate with the ground surface through a drainage shaft or an adit (Fig. 317). In addition drive-pipe and through-filters are employed. These are short perforated pipes with nozzles inserted into 40-65-mm holes drilled from the underground openings to aquifers. Through-filters are inserted into boreholes 150-300 mm in diameter drilled from the ground surface down to the drain workings. The space between the borehole walls and the filtering pipe is filled with gravel. Through-filters are used in cover rock strata containing a series of aquifers which are separated from one another by layers of water-impermeable rock.

With the underground drainage method pumping plants are set up in underground pump rooms situated near the drainage shaft. The water runs down to the header along underground drain openings (Fig. 317).

Pumping plants for preliminary underground dewatering are usually also used for drainage during the exploitation of the open-pit. Then there is no need of surface drainage from the quarry floor, for the water flowing to the bottom of the open-pit goes down to an underground header. Underground drainage does not interfere with the pit operation and is more reliable, this being of importance with a large water inflow, which in some pits reaches 500 m^3/hr and more.

CHAPTER XI

ORGANIZATION OF OPENCAST WORK AND ITS TECHNICAL AND ECONOMIC CHARACTERISTICS

11-1. GENERAL PATTERN AND METHODS OF MINING OPERATIONS

Removal of Cover Rocks and Driving of Ingoing Trenches

After all the necessary work in connection with the construction of the open-pit (setting up of communication and electric power supply lines, securing water supply, building warehouse facilities, industrial plants and structures, change houses, etc.) has been completed, mining operations in the open-pit begin with the stripping of overburden or removal of cover rocks.

In the majority of open-pits the overburden is removed by means of power shovels. To this end a trench is driven, then the overburden is removed by making parallel cuts on one or both sides of the trench. Conditions permitting, the overburden can be stripped by a dragline or a multibucket excavator.

The overburden can also be stripped by using drag or wheeled scrapers. Scrapers are widely used in open-pits of small production capacity, where the use of heavy-duty excavators and transport facilities is inexpedient.

Under certain conditions (soft ground, sufficient supply of water) stripping may be done by hydraulicking, which is very efficient and is employed at a number of Soviet open-pits. Hydraulicking includes the following operations: washing of the ground at the face, moving of the washed-out earth over the necessary distance, and construction of a hydraulic dump. Since these operations are simple, labour efficiency high and initial outlays for the purchase of equipment small, the use of this method in stripping of overburden and soft ground is progressive and very promising.

After complete or partial stripping of overburden or removal of cover rocks has been done permanent ingoing trenches are driven in accordance with the adopted method of opening-up the open-pit. The driving of ingoing trenches and removal of overburden or overlying cover rocks prior to the actual exploitation of the pit are financed from the capital investments.

Development Work and Actual Mining

Development in an open-pit consists in the driving of working trenches and the excavation of barren rock on production horizons. Stoping consists in the extraction of the mineral. Development and stoping are started after the driving of ingoing trenches in conformity with the time schedule for the working of the deposit. One must consider the possibility and expedience of synchronizing different kinds of mining operations on one or several horizons, especially in deposits of considerable size. Establishing the necessary sequence or synchronization of various operations and their scope in individual years is the paramount problem, which is solved in the course of drawing up time-schedules for open work. Prior to the stripping of waste or extraction of the mineral on a particular horizon working trenches should be driven. The waste or mineral is extracted by individual strips, first from the working trench and then from the corresponding sections of mining areas.

In the practice of opencast mining the direction of advance of the working face front with respect to the working trench is parallel, fan-shaped or combined. When choosing the direction of advance consideration should be given to the elevation marks of the mineral floor (in single-bench extraction), variations in the thickness of the overburden and the mineral deposit, and the conditions of drainage.

The main disadvantage of parallel advance of the working face front with the use of rail transport is the necessity of installing turnout switches when setting up operation in a new cut. Its basic merit consists in the constant width and length of cuts as distinct from fan-shaped advance, where they are variable. The advantage of fan-shaped advance is the elimination of the need for turnout switches when starting a fresh cut.

Because of the shortcomings inherent in fan-shaped advance of the working-face front (the variable width and length of cuts complicates the organization and technology of mining operations) it is rarely used. The method of parallel advance is the most popular. The basic mining operations in an open-pit during the stripping of waste and excavation of the mineral have been described previously.

11-2. BACKGROUNDS FOR DRAWING UP A TIME-SCHEDULE OF OPENCAST WORK

By a time-schedule is meant the distribution in space and time of stripping and stoping operations within the boundaries of an open-pit field, which assures the systematic, safe and efficient working of the deposit throughout the life of the open-pit. To draw up a time-schedule one must possess the following data:

reserves of the mineral in the open-pit field by individual horizons, with special reference to bench elevation marks;

depth of open work and basic quarry parameters;

adopted methods of opening-up and development of the open-pit field and the direction of progress of stoping;

open-pit plans by individual horizons with plotted lines of the mining front which should ensure the opening-up and development of the horizons.

This information serves as a basis for investigation into the cycle of mining operations and for constructing a chart showing the rela-

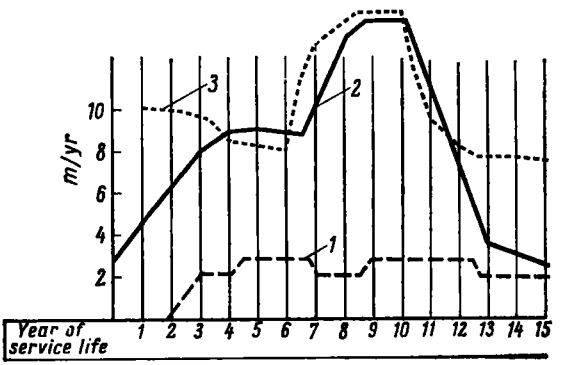


Fig. 318. Time-schedule for a mining cycle:
1—volume of mineral; *2*—volume of overburden; *3*—progress of mining in depth

tionship between the growing volumes of the stripped material V and the growing volumes of the mined ore P:

$$V = f(P)$$

In analysing this formula the stripping-to-ore running ratios are averaged out and the main stages of deploying the mining operations determined, viz. the construction period, the period of attaining the planned production capacity, the period of normal exploitation, and the period of reclamation. An example of a time-schedule for the mining cycle is given in Fig. 318 [1].

In some cases it may be found advisable to work the open-pit in several stages, this allowing the scope of initial stripping to be cut down sharply. On deciding upon the main stages in open-pit operation the excavators are spotted on the horizons, the priori-

[1] V. Rzhevsky defines the mining cycle as an established sequence of stripping and actual mining within the quarry's field, which should ensure the systematic, safe and efficient working of a deposit throughout the life of the open-pit.

ties and dates of the beginning of their exploitation are determined and so are the efficiency of each excavator and the total amounts of the bulk to be mined in each year. The placement of excavators usually introduces some corrections into the previously drawn up operational programs for individual horizons. The time-schedule is prepared in the form of a table which lists the years of operation, the horizons, the numerical designations of the excavators and tonnages handled by them, and encloses mining operation programs for individual horizons in graphic form. The general time-schedule for the progress of mining at an enterprise serves as a groundwork for drawing up monthly, quarterly and annual operative programs.

11-3. GEOLOGICAL AND MINE-SURVEYING SERVICE IN DRILLING, BLASTING AND EXCAVATION

Adequate organization of geological and mine-surveying service in drilling, blasting and excavation is a matter of great importance, since the ultimate results of the work largely depend on it.

Servicing of Drilling and Blasting Operations

On the basis of the standard blast-hole pattern the Mine-Surveying Department maps out the mouths of blast-holes all over

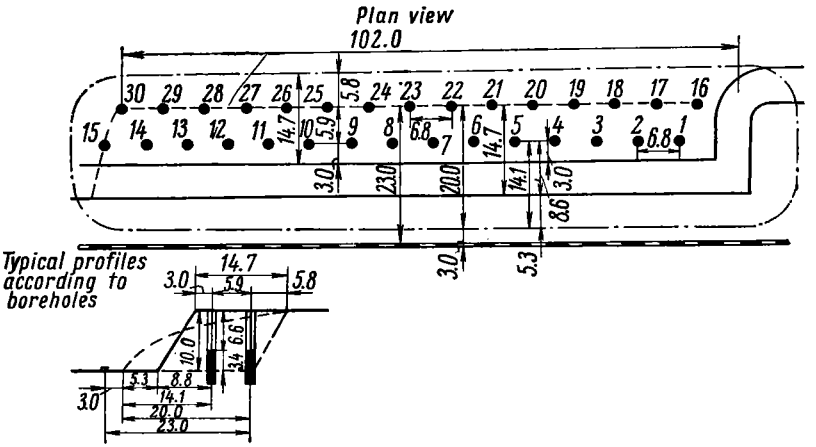


Fig. 319. Sketch map of drilling-and-blasting operations

the producing section. If any changes in the physical and mechanical properties of the traversed rocks are discovered in the course of drilling the Blasting Department introduces the corresponding

corrections into the location of the blast-holes. Once drilling is terminated the holes are checked for depth. Geologists then compile detailed geological sections, indicating the kinds of rocks in accordance with the specimens obtained in the course of drilling. On the basis of precise profiles the Blasting Department calculates exact charges for each blast-hole. When drilling and blasting in open-pits special technological maps or specifications are consulted (Fig. 319).

Exploratory Assaying

This mainly consists in sampling the borehole slime at definite space intervals. Chemical and mineralogical analyses of specimens furnish a background for preparing an assay map according to which selective mining of the ore is done. Figure 320 illustrates an example of an assay map (according to ore grades), serving as a guide for blasting operations at one of the Soviet copper ore open-pits. To ensure a better coverage of mining operations a geologist is assigned to each producing section, whose duty it is to take care of all geologizing. Specifications are issued for all batches of ore extracted in the open-pit with the quality of the ore indicated in them.

Maps for mining operations are drawn up to scales of 1 : 500, 1 : 1000, 1 : 2000. For the sake of clarity summary bench-wise maps of the entire pit are drawn up to a scale of 1 : 2000, on which the grades of ore, the lines of working faces and the projected pit contours are shown. On assay maps and those indicating sites of spalling in the faces the latter and boreholes and existing survey pegs are plotted. Major measurements of excavated ground and mineral are made once a month.

11-4. WORKING TIME AND CYCLIC ORGANIZATION OF MINING OPERATIONS

Cycle of Open-Pit Operation

An open-pit may operate on a continuous or intermittent cycle. In the continuous cycle the pit is busy 357 days a year, in the intermittent cycle with one day off per week 305 days, and with two days off 253 days a year. In determining the annual pattern of activity of an open-pit it is necessary to take into account the operation of related enterprises, the capacity of intermediate stock pile sites, and the climatic conditions. Two-shift mining should be regarded as the most practical, a third shift being used for blasting, overhauling of equipment and various ancillary operations.

The number of working days in a year for the principal pieces of equipment should be: for power shovels 290-300, for stripping excavators from 240 to 268 (in central regions of the USSR). To ensure

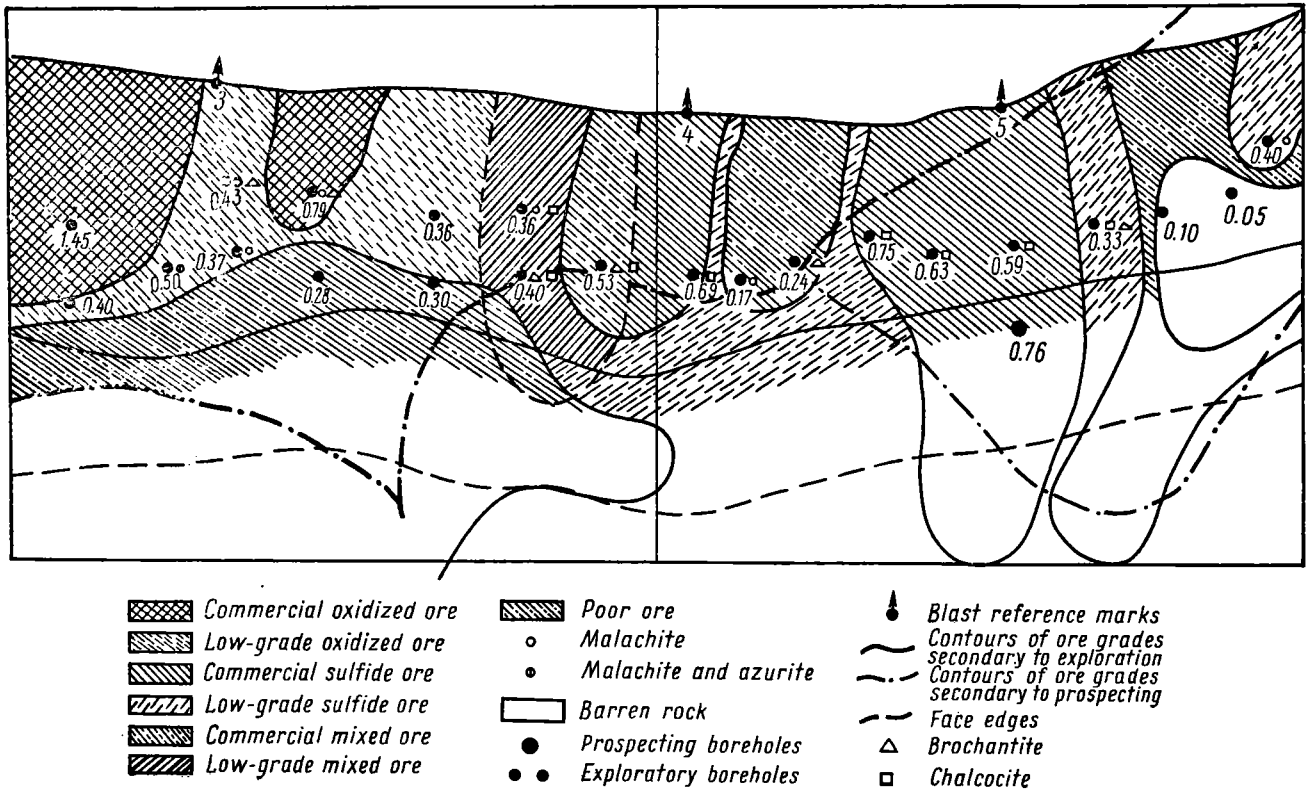


Fig. 320. Assay map for blasting operations

the fulfilment of the annual production program it is necessary to have stand-by equipment and machinery comprising 20-25% of the total number in actual operation.

Cyclic Mining

Work in an open-pit should be conducted in accordance with approved schedules. As stated earlier, work on schedule implies

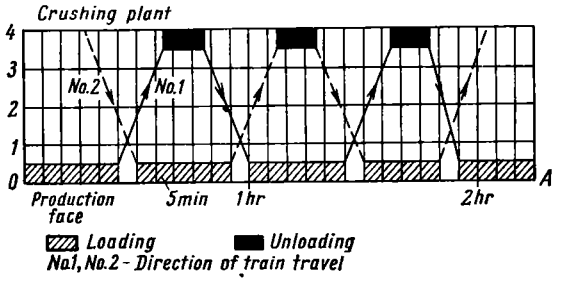


Fig. 321. Operational schedule for two trains and an excavator

carrying out mining operations within stipulated time limits and in a definite sequence. At Soviet open-pits most of the stripping

Operations	Per-shift efficiency	Days in 10-day periods and shifts									
		1	2	3	4	5	6	7	8	9	10
		1 2 3	1 2 3	1 2 3	1 2 3	1 2 3	1 2 3	1 2 3	1 2 3	1 2 3	1 2 3
Churn drilling (rig No...)											
Power-shovel loading (excavator No...)											
Transportation (train No..)											
Track shifting at face											
Track shifting at dump sites Dump No.1 Dump No.2											
Charging and firing deep blastholes											

Fig. 322. Composite progress chart for one production face and dumps servicing it

and stoping operations are done with the aid of drill rigs, excavators, mechanical transport facilities, and other equipment, i.e. all operations are fully mechanized. The basic and most conve-

nient efficiency indicator in open work is the vehicle turnover and the time utilization factor of digging equipment. An excavator and trains assigned to it operate according to schedule, as a rule (Fig. 321). The same refers to other operations at the production face.

Schedules for individual types of work serve as a basis for compiling the so-called composite progress chart for a face, a producing section or the open-pit as a whole (Figs. 322, 323). Operational schedules and progress charts should enclose the necessary information as to the expected output of the principal pieces of equipment and machinery for each day of the month.

The observance of operational schedules should be kept under strict control. To this end "work done" charts are compiled, which are then analysed.

The basic measures to ensure operation on schedule are: rigid work discipline, timely execution of running, planned and preventive repairs and overhaul of the mechanisms, the availability of the necessary reserve equipment and stand-by faces in the open-pit; timely supply of materials, tools and spare parts.

Organization of Work During Open-Pit Construction

In the foregoing text we have reviewed the basic problems involved in the organization of work during the exploitation of an open-pit. It is also very important to maintain high standards of operation during open-pit construction in line with a previously prepared time-schedule of early development and construction work. This schedule should include: a list and the scope of major operations on each horizon (driving of permanent trenches, through-cuts, working trenches and extension of benches), a list of excavators in operation (types and conventional numerical designations), the places and dates of their operation (years, months, shifts). The time-schedule for early development and construction must also contain data on the overall volume of major permanent excavations, duty-cycle of excavators (seasonal, all-year-round) and the construction time of the open-pit before it is put in exploitation (usually not more than 3 years). The time-schedule must enclose a plan of the open-pit as of the date of its commissioning, with trenches, benches, basic quarry dimensions and length of trenches indicated on it. A typical form for such a time-schedule is shown in Fig. 324.

During open-pit construction numerous and complex operations (excavation, building, assembly) are accomplished, most of which are interrelated. With the conventional system of planning it is difficult to establish the logical sequence and interdependence of individual operations. To elaborate an optimal overall program for the operations during open-pit construction the cross-flow method

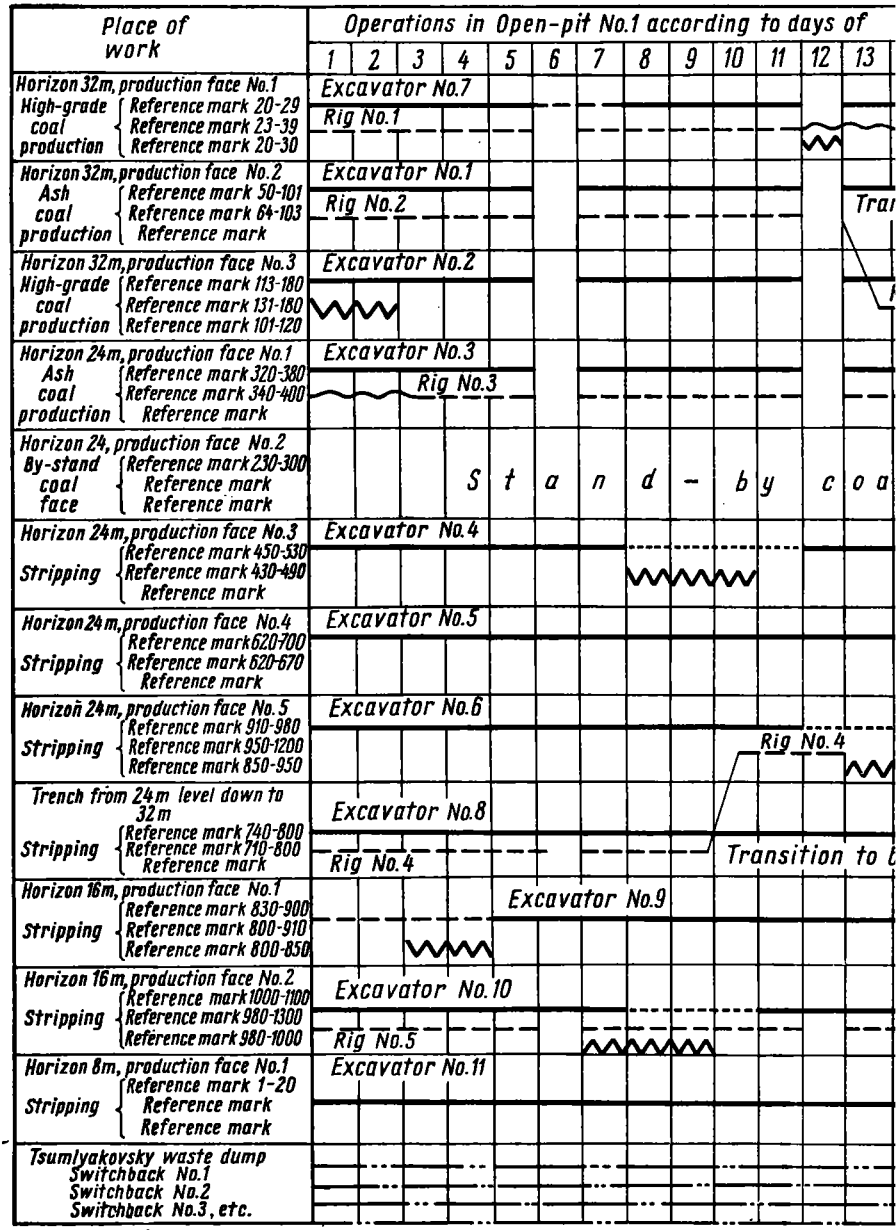
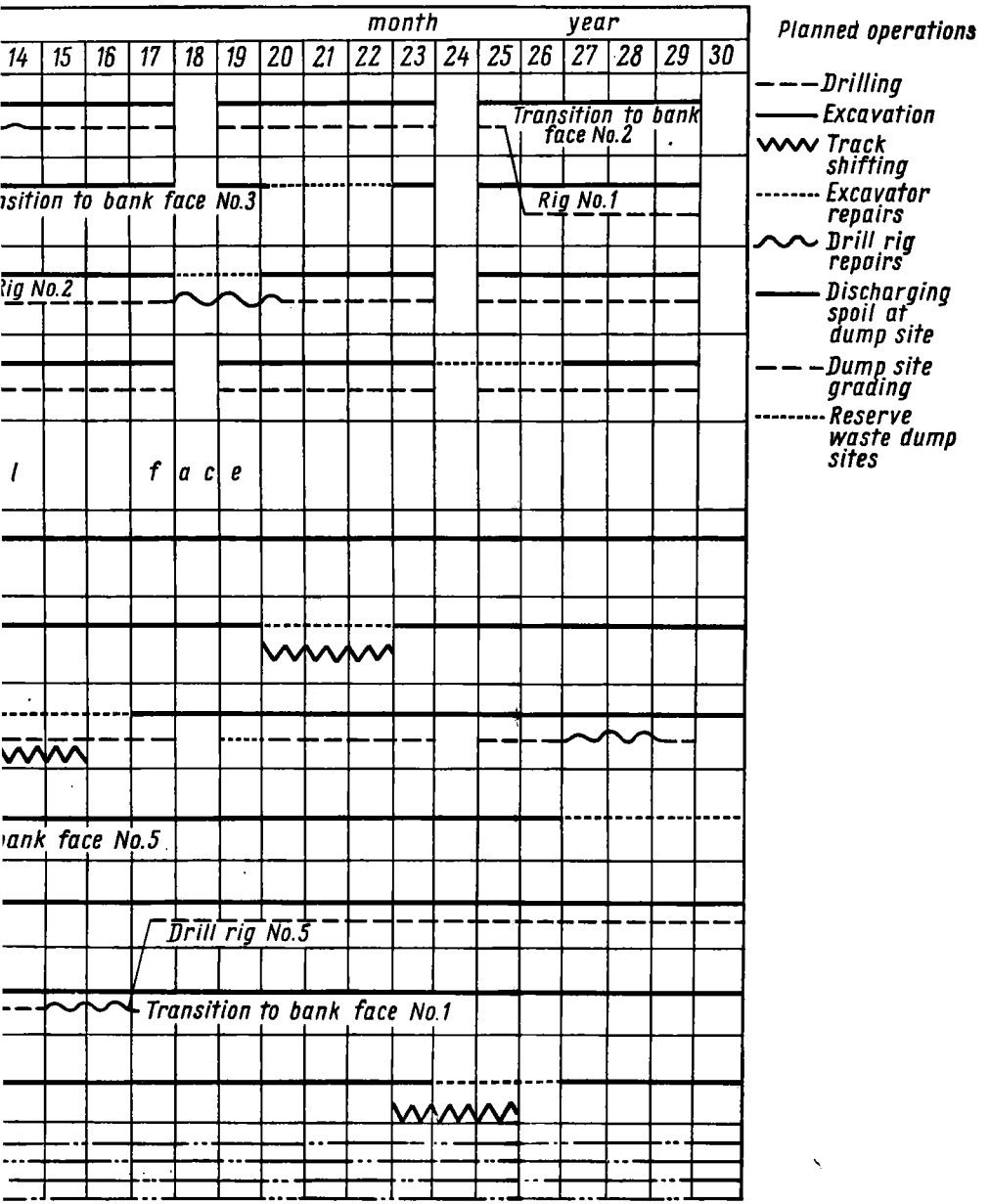


Fig. 323. Overall monthly operational s



chedule for an open-pit during its exploitation

Designation of operation	Volume, thous.cu m	Excavator type and No.	1st year												2nd year												3d year		
			Months												Months														
			1	2	3	4	5	6	7	8	9	10	11	12	1	2	3	4	5	6	7	8	9	10	11	12	1	2	3
Driving outside permanent opening trench i=... ‰ L=... m ; H=... m		ЭКГ-5 No.1																											
Drivage of working trench at level... m L=... m ; B=... m		$R_{load}\frac{300}{3.8}$ 24 No.2																											
Extension of bench berm at level... m etc		$R_{load}\frac{300}{3.8}$ 24 No.2																											

Fig. 324. A form for an open-pit construction time-schedule

of planning and control is applied, with the use of electronic computers when necessary. A cross-flow diagram enables one to establish the list of individual tasks and the time periods needed for completing them, which determine the total time necessary to accomplish the entire set of operations in the open-pit. The route on the diagram which these operations follow is called critical. The drawing up of a summary optimal cross-flow chart should be preceded

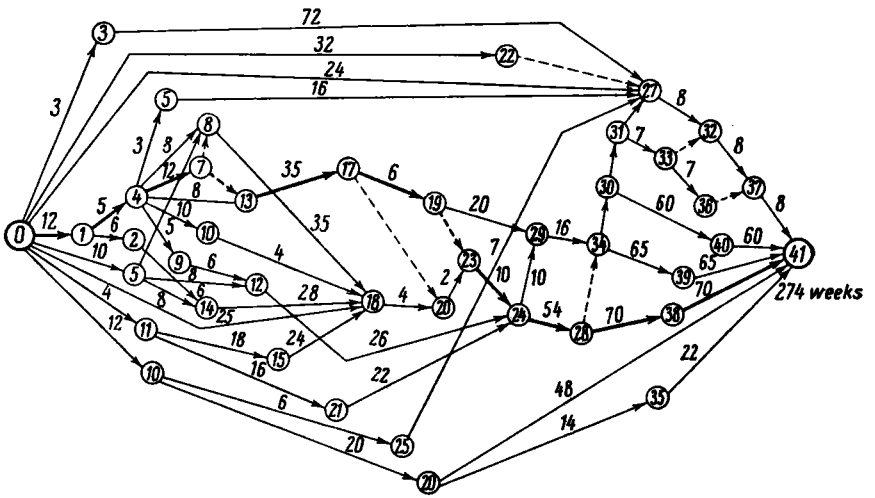


Fig. 325. Summary optimal cross-flow work schedule (PERT chart) for open-pit construction

0-1-4-7-13-17-19-23-24-28-38-41 critical route = 274 weeks

by the following: making a list of and assigning code numbers to the preceding and succeeding events, setting the time for individual types of operations (usually in terms of weeks), elaboration (when necessary) of specific optimal cross-flow charts for individual types of complex operations (for example, assembly of a rotor excavator). In especially difficult cases a primary cross-flow chart is first compiled, which is then adjusted with a view to cutting down the periods envisaged for the execution of individual tasks; then a final summary cross-flow chart is prepared in which the critical route and its duration are indicated. Figure 325 illustrates a summary optimal cross-flow chart for the construction of one of the open-pits, which includes 41 events and a critical route shown as a solid line whose length represents a period of 274 weeks.

11-5. APPROXIMATE TECHNICAL AND ECONOMIC CHARACTERISTICS OF OPENCAST MINING

1. Marginal stripping ratio from 3-5 to 15-20 m^3/t, depending upon the mining conditions and the equipment used.

2. Characteristics of drilling and blasting operations:

efficiency of a churn-drill rig for a blast-hole 200 mm in diameter—from 2 m/shift in very strong rocks to 25-30 m/shift in soft ground;

average ammonite consumption in coal open-cuts: 0.15-0.20 kg per 1 ton of coal, 0.10-0.25 kg per 1 cu m of country rock;

average ammonite consumption in ore open-pits in primary blasting from 0.3 to 0.6 kg/m^3 of rock or mineral, depending upon hardness; in secondary blasting—8-10% of the amount used in primary blasting;

yield of ore per 1 metre of churn-drill hole—40 cu m on the average for all ore open-pits of the USSR and 80 cu m for the best ones.

3. Characteristics of excavator operation:

efficiency of an electric power shovel per 1 cu m of bucket capacity: for soft ground (sandstone, top soil) 200-350 thous. cu m of ground per year; for clayey compact ground 250-300 thous. cu m; for semi-rocky ground requiring partial blasting 200-250 thous. cu m; for rocky ground with the use of blasting 150-190 thous. cu m;

the output of a dragline may be estimated tentatively on the basis of the efficiency of a power shovel by adopting the corresponding coefficients: 0.82 for soft ground, 0.80 for firm ground, 0.77 for semi-rocky ground, and 0.64 for rocky ground;

the efficiency of a multibucket excavator per 100 litres of bucket capacity is 100 cu m of ground per hour of total digging time.

4. Characteristics of haulage operations:

efficiency of an electric locomotive with an adhesion weight of 80-100 tons—2,500-3,000 ton-km per shift (with a one-way haul of 3-3.5 km);

efficiency of a motor dump car per 1 ton of its load-carrying capacity—90-100 ton-km per shift (with a one-way haul of 2.5-3.0 km);

monthly efficiency of a wheeled scraper per 1 cu m of capacity of the total available scraper fleet—650-750 cu m (with a slushing distance of 300 m);

electric power consumption with electromotive haulage—0.15-0.20 kW-hr per 1 ton-km;

fuel consumption by motor transport per 100 effective ton-km—up to 10 litres with ordinary engines and 5-6 litres with diesel engines.

5. General characteristics of opencasting:

average rate of advance of the working front depends upon the scope of operations, capacity of excavators, type of transport, pro-

perties of ground (in rocky ground—up to 100 m/yr, in soft ground—up to 200-250 m/yr);

average annual progress in depth is contingent upon the same conditions as above and may be set at 11-18 m in semi-rocky ground and 9—15 m in rocky ground with the use of r.r. cars. With motor haulage these rates can be increased.

Initial outlays per 1 ton of annually excavated bulk amount to approximately 2.5-2.0 roubles for big open-pits (with a production capacity of up to 5 million tons of muck), 2.5-4.0 roubles for medium-sized ones, and 4.5-8.0 roubles for small ones;

output per open-pit worker ranges from 5-10 to 50-100 cu m of mined bulk per shift. The smaller figures imply strong rock, a low production capacity of the open-pit and the use of railway haulage, while the higher figures imply excavation with overcasting of soft ground and the working of large-capacity open-pits.

The total cost of excavating 1 cu m of ground is 0.15-0.3 rouble with overcasting of soft rock and 0.4-1.0 rouble with railway haulage. Mining cost (less excavation of barren rock): 0.6-1.2 roubles per cu m for coal and 0.8-1.8 roubles per cu m for ore:

percentage distribution of the total cost of mining the bulk, according to expenditure items (with haulage of excavated strong rock):

Drilling and blasting	15-25
Digging	10-15
Haulage	40-50
Waste disposal and stacking	15-20

soft ground requires no drilling and blasting, and this changes the percentage distribution of other expenses;

total mining cost charged against 1 ton of the mineral varies widely depending upon the scope of stone work (stripping ratio) and the open-pit production capacity; overall cost of mining the mineral, including stone work, may be estimated tentatively in each individual case proceeding from the amount of waste per 1 ton of the mineral and the net cost of excavating 1 cu m of muck; in the presence of cover rocks account must also be taken of expenses incurred in removing them;

total electric power consumption per 1 cu m of mined bulk amounts to 2-5 kW-hr;

exploitation losses in mining, loading and transportation constitute from 2 to 5.0% in the absence of gangue intercalations; when such intercalations are present the losses rise correspondingly and may reach 8-10% under extremely complex mining and geological conditions; losses in safety pillars and berms are estimated in each individual instance;

dilution of the mineral is 2-3% in the absence of gangue intercalations, while in their presence dilution should be determined in each individual case.

Technical and Economic Characteristics of New Soviet High-Capacity Open-Pits (Estimate)

As stated earlier, plans have been prepared in the USSR for the construction of new highly-mechanized coal open-cuts, each with a projected annual output of 40-60 million tons of coal. Initial investment per 1 ton of annual production at these open-pits will amount to 3.4-2.17 roubles. Output per manshift will be 60-90 tons of coal, the mining cost of coal comprising 0.36-0.29 rouble. Labour efficiency here will be about four times as high as at US coal open-cuts.

11-6. BASIC MEASURES TO RAISE THE EFFICIENCY OF OPEN WORK

Due to all-round mechanization, the utilization of highly efficient equipment and improved organization of work it became possible to extend the scope of opencast mining in the USSR and increase the marginal stripping ratio (in particularly favourable conditions of working soft ground) to 15-20 cu m per ton.

Experience gained with opencast mining in the USSR during the last decade shows that the efficiency of opencasting mineral deposits can be considerably increased, provided the necessary technical and organizational measures are carried out.

The basic measures to raise the efficiency of open work are as follows:

1. Carrying out stripping operations with the use of heavy-duty excavators and large-capacity wheeled scrapers pulled by caterpillar-mounted tractors, and hydraulic removal of overburden.
2. Working of deposits with the use (under favourable conditions) of side-casting and overcasting or systems involving haulage of waste over short distances to inside waste-dumps.
3. Utilization of mining, transport and ancillary equipment of compatible types and power ratings.
4. Mechanization of operations with the use of technically perfect machinery—powerful rotary and thermal drill rigs, electric power shovels with a bucket capacity from 3-4 to 7-8 cu m, powerful electric and diesel engine locomotives, motor dump cars carrying loads of 10-40 tons and more, and also conveyer haulage.

5. Operating in accordance with approved time-schedules, cross-flow labour and time planning charts and introducing progressive methods of work.

6. Efficient organization of the mine-surveying and geological services and making them responsible for control of the quality of the mineral and the implementation of measures to cut down losses and dilution.

The basic measures to reduce qualitative and quantitative losses of mineral during opencast mining are: setting the correct high-wall, bank and berm angles so as to prevent collapses, downfalls and slides; recovery of all abandoned pillars of mineral during second mining; complete extraction of mineral in the back and bottom of the deposit (this requires regular clean-ups with tractor planers or skimmers); regular assaying of the mineral in the producing section of the deposit.

7. Selective mining with the use of the appropriate equipment in extracting mineral of different grades, or in the presence of considerable gangue intercalations. In selective extraction the bench height, the bucket capacity of the power shovel and the spacing of blast-holes should be in line with the requirements imposed on this mode of mining. Specifications for the blocks must be prepared in good time, the muck should be rent from the solid mass and loosened without too much scatter, the front of stoping operations enlarged and carry-over stocks of different grades of mineral built up in piles and on the surface of the open-pit.

8. Improved technique and organization of secondary blasting of oversize in extracting hard rocks.

9. Timely exploration and drainage of sections to be mined. This largely affects the safety and efficiency of open work (losses and dilution, quality of mineral, stability of open-pit high-walls and banks, the scope of stone work, etc.).

10. Mechanization of waste disposal with the use of spreader ploughs, excavators and track shifters.

11. Extensive utilization of earth-moving machines designed for mechanization of auxiliary operations (tractor ploughs for cleaning up the mineral and waste, wheeled scrapers and tractor ploughs for planing and grading of work areas, cranes for track shifting, and other machinery).

A further improvement in the mining processes and in the organization of stripping and mineral-extraction work in open-pits, and also large-scale progress of research, are major steps towards considerable increase in the production of minerals by the opencast method.

II. SURFACE MINING OF PLACER DEPOSITS

General. In the USSR the surface method is practised in the working of gold-bearing, platinum, diamond, tin, ilmenite and other placers containing rare metals. The principles of mining gold-bearing placers are applicable to the working of other placers as well.

Gold placer deposits vary in their size. Their length may reach a few kilometres, their width varying from 40 to 1,000 m. The thickness of placers mined by the surface method may reach 50 m.

The small content of gold in the sands and its non-uniform distribution necessitate thorough and regular assaying of the sands. Gold-bearing sands often carry valuable accessory minerals (platinum, cassiterite, precious stones, rare metal minerals, etc.), which are often caught along with gold. In assaying a placer sufficient attention should be given to minerals occurring together with gold, and the commercial value of sands should be established on the basis of the total content of useful components.

The working of placers is preceded by prospecting by sinking test-pits and boring. Test-pits and boreholes are arranged in rows with a definite spacing depending upon the nature of the placer and the category of reserves under exploration. The following data are needed to choose a rational method of mining and to prepare the project for the exploitation of the deposit:

general economic and geographical information covering the deposit and the area adjacent to it, ways of communication, sources of water and electric power supply;

brief geological characteristics of the placer;

cross-sections of individual test-pits and boreholes accompanied by data on the thickness of the rock strata, the contents of useful constituents, the strength and toughness of the ground, inclusions of hard rock, the coefficient of expansion and specific weight of the rocks;

size grading of individual kinds of rock;

characteristics of the underlying bedrock (surface relief, strength, degree and depth of ground breakage);

the nature of the metal contained in the placer: its specific weight and fineness, size grading, susceptibility to amalgamation in the case of gold, magnetic properties of platinum, cassiterite and wolframite; metal reserves in individual areas of the placer;

monthly water discharge in the stream (river, etc.) for a period of many years;

stream velocity and its variations, information on the width and depth of the stream;

monthly data on precipitations, winds, temperatures, snow cover and depth of freezing;

topographic map to a scale of 1 : 50000-1 : 100000;

plan of the placer to a scale of 1 : 1000-1 : 2000 indicating its contour and prospecting lines, boreholes showing metal content, and all existing structures, workings, dump sites, permafrost areas, swamps and water reservoirs;

placer cross-sections along prospecting lines indicating the water level in the stream, the subsurface water table and elevation marks of permafrost, with sand and peat strata singled out.

Once exploration is completed the placer is worked either by the surface or the underground method, depending upon their expedience, which should be ascertained by technical and economic calculations.

In surface mining of a placer distinction is made between the preparation of the ground for mining and the extraction of sands. The former stage includes removal of the vegetative cover and setting up a water supply system, dewatering of the ground, securing water needed for washing sands at the gold-washing plant. If a stream or a small river flows across the area of the future goldfields it is diverted by means of a special water-diversion channel or a line of flumes of appropriate size. To protect the plot from flooding during spring high water it is fenced off on the side of the stream by a longitudinal dam with the necessary additional ditches. The plot is drained by means of a permanent outfall ditch carried from the lower part of the placer field on the floor of the gold-bearing stratum. Washeries are located near the stream or the outfall ditch. The preparation of the placer area for mining is accompanied by the construction of a settlement for workers and employees, erection of light industrial structures and plants, and by clearing the ground for the peat dump sites.

Extraction of peat is done by benches of different height, depending upon the system of mining employed. The general sequence of stripping operations is the same as in the working of open-pits. The removal of peat capping is often done in winter months, when frozen sands are not extracted because of considerable difficulties involved in their washing. Owing to their small thickness sand strata are commonly taken out in a single bench. The mining of sands is at its height in the warm season, when gold washeries operate most effectively. In surface mining placers are worked by excavators, dredges, hydraulic equipment and bulldozers with scraper trailers.

Mining by excavators has been discussed above. Now we will consider the other three methods of placer mining.

CHAPTER XII

DREDGING OF PLACERS

Construction of the Dredge

The dredge is a floating digger intended for mining sands in pay placers which contain commercial amounts of useful components and washing them to separate metals or minerals from waste (Figs. 326, 327). The basic parts of the dredge are: a pontoon, a dredging mechanism, structures supporting the working equipment and holding it in place, concentration (washing) equipment, and power plants. Dredges are classified as continuously (multibucket) and intermittently (single-bucket) acting depending upon the design of the digging mechanism. The latter type has found little practical use. Electrically and diesel-powered dredges are the most popular. As concerns the construction of the dredging mechanism (which mainly consists of a chain with scoops attached to it), multibucket dredges are available in two types. In the first type the buckets are directly coupled to one another, forming a continuous chain, whereas in the second type idler chain links are inserted between individual buckets. Dredges of the first type can be used under varying working conditions, and therefore they find more extensive application. When in operation, dredges are moved along with the aid of spuds or lines.

Heavy and strong spuds placed at the stern of the dredge hold it in position during operation. Spuds represent riveted steel beams lifted by pulley blocks, which are fastened to the rear gantry. The lower end of the spud is provided with a cast steel shoe, which penetrates into the bottom of the pond. When operating at the face the dredge leans against one of the spuds, revolving around it by manipulating side lines. At this time the second spud is raised. On completion of a cut and cleaning up the underlying bedrock the dredge is pulled one step further. The travel of a dredge with the use of spuds is shown in Fig. 328. Two lines are sufficient to make such a dredge swing around the spud axis.

Line-operated dredges are held in place by five long cables with one end wound on the drum of a winch set up in the dredge, the

other being firmly anchored either on the bank or on the bottom of the pond. When digging, the dredge slowly swings in a pendulum-like fashion around the point where the head line is fixed.

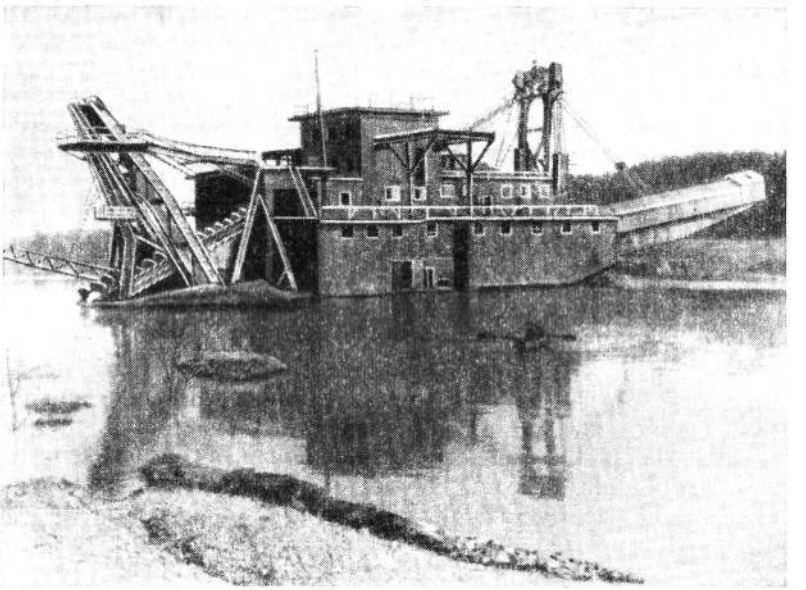


Fig. 326. General view of a 380-litre capacity dredge

To accomplish this, the lines on one side should be tightened and those on the other side paid out (Fig. 328).

Below we give a short description of the basic parts of the dredge (Fig. 327).

The pontoon is a rectangular box-like hull made of steel or wood and divided inside by water-tight bulkheads; it has at its bow end an opening for the digging ladder extending along the pontoon over 1/3-1/2 of its length. Thus the digging ladder can be shifted to change the digging depth. The bow of the pontoon has chamfered sides so as to facilitate manoeuvring the dredge during its operation. The main requirements imposed on the pontoon are its strength and rigidity. The length of the pontoon is indicated in Table 43. The average draught varies from 1.5 to 2.5 m.

Supporting structures are erected on the pontoon deck; these are trusses, uprights, braces, jibs, etc. The most important of them are the main frame, the bow and stern gantries.

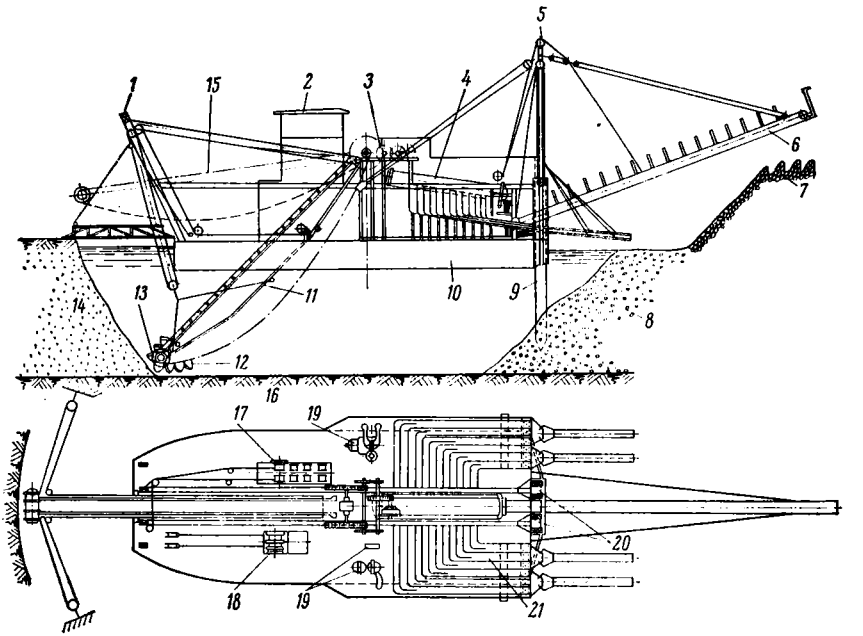


Fig. 327. Basic parts of a dredge:

1—bow gantry; *2*—dredger's cabin; *3*—bucket chain drive; *4*—trommel; *5*—rear gantry; *6*—conveyer stacker; *7*—coarse-tailings dump; *8*—fine-tailings dump; *9*—spud; *10*—pontoon; *11*—bucket ladder; *12*—bucket; *13*—lower tumbler; *14*—placer; *15*—top portion of bucket ladder; *16*—bedrock; *17*—main winch; *18*—auxiliary winch for lifting bucket ladder; *19*—pump; *20*—spud; *21*—sluices

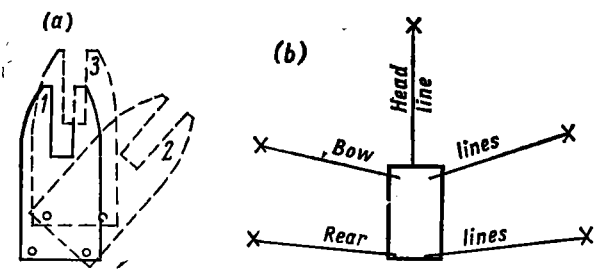


Fig. 328. Dredge travel diagram:

a—with the use of spuds; *b*—with the use of lines; *1*—initial position of dredge; *2*, *3*—dredge stepping positions

The main or central frame supports the top end of the digging ladder with its drive. It is placed approximately in the middle of the pontoon. The bow gantry holds the lower end of the bucket frame

in place by means of steel cables suspended from it. The stern gantry carries the conveyer frame, tail sluices and spuds. The gantries are made of wood or steel.

The dredging ladder consists of a bucket frame, a bucket chain and two tumblers placed at the top and bottom ends of the bucket frame, respectively. The latter is made of steel beams held together by cross-pieces. The length of the frame is chosen to suit the digging depth. The front face of the bucket frame is provided with evenly spaced rollers on which the loaded portion of the bucket chain moves. The bucket consists of a bottom and a body, whose cutting edge is fitted with a lip. Buckets, and especially their cutting edges are manufactured of the best grades of special steel. The bucket walls are 6-10 mm thick, the lip up to 30 mm thick. The bucket has a capacity of 50-600 litres.

Dredges are equipped with a washery incorporating a barrel, jiggers and sluices. The material scooped up by buckets, having passed through grizzlies (to screen out big boulders) and a feed chute, goes into the barrel. The purpose of the barrel is to loosen the material, to separate boulders and large pebbles and to supply sluices with material containing grains not larger than 18 mm. Dredges usually have open cylindrical barrels, measuring 3 m in diameter and 17 m in length and set up at an angle of about 6° to the horizontal. The barrel sides are made of metal sheets riveted to the framework and having round holes, whose diameter increases from 8 mm in the upper end to 18 mm in the lower end. The barrel is set in motion by friction rollers. The water needed for washing the material is supplied to the barrel under pressure through a special spray pipe. Underneath the barrel there is a distributing arrangement ensuring a uniform flow of the material screened through the barrel holes to both sides of the sluice.

The purpose of the sluices is to trap grains of gold freed from sand and clay. Sluices are troughs 0.81 m wide set up with a slope of 0.100-0.125 and provided with stencils for trapping grains of gold. After leaving the distributing arrangement the material goes first to the lateral and then to the longitudinal sluices. The washed material (ettle) is dumped at some distance behind the dredge. Coarse oversize (pebble, boulders) is transported to the dump site by a belt-conveyer stacker.

Table 43 lists characteristics of some dredges manufactured in the USSR.

The great variety of dredge types may be attributed to the fact that they have to be designed for mining placers with different reserves of the mineral and depth of occurrence. Dredges for working small placers are equipped with buckets of up to 50-litre capacity and have wooden pontoons.

Table 43

Characteristics of Soviet-Manufactured Dredges

Characteristics	Irkutsk Plant (1)		Perm Plant		Irkutsk Plant (2)
Bucket capacity, litres . .	150	250	380	380	600
Digging rate per minute . .	22	20-34	22	22	20
Maximum depth of digging under water, m	9.3	12	11	30	50
Bucket frame length, m . .	23.5	—	29.5	58	—
Pontoon material			Steel		
Length of pontoon, m . .	32.5	42.8	45.5	62.0	102.0
Pontoon draught, m . . .	1.8	2.0	2.6	2.5	—
Spud length, m	13.9	18.1	19.6	26.0	—
Stacker length, m	30.3	33.5	47.5	70.0	—
Weight of dredge, tons . .	720	124	1,950	3,370	7,500

CONDITIONS FOR THE USE OF DREDGES, THE TECHNOLOGY AND ORGANIZATION OF DREDGING

Dredges are employed under definite conditions of mining the major of which are the following: the depth of the placer occurrence should correspond to the digging depth of the dredge adopted for the purpose; the presence of a water basin or the possibility of making a pond; the reserves of pay sands should be sufficient to ensure the operation of the dredge for a period of not less than 10 years; the placer must not abound in big boulders exceeding 1/3 of the dredge bucket in size; the top of the bedrock should lend itself to the breaking action of the buckets, because otherwise the losses of metal may be high.

For large- and medium-size dredges water discharge in the placer valley should be 100-150 litres/sec. With a smaller water discharge special measures should be taken to reduce the water runoff or to supply water from neighbouring valleys. In areas deficient in water, particularly those with a cold climate, water is sometimes supplied over a distance of 20-30 km.

Opening-up a placer. In dredging, the opening-up of a placer has specific features, since the dredge is a floating unit and yields a finished product without any need for sand transportation.

There are three basic methods of opening-up: by pits, with deep digging out of the bedrock, and with the aid of dams.

In the first method the dredge is placed in a pit (Fig. 329). The location of the pontoon bottom depends on the water level in the pit. The pit is excavated to a size ensuring free movement of

the dredge during actual mining. For example, to accommodate a 250-litre capacity dredge the pit must measure about 50 × 70 m. In this method of opening-up the costs sustained are moderate. The dredge cannot come near an area with small underwater depth, or a high underwater bank, and is forced to do bulk mining of both the sands and peat of the placer.

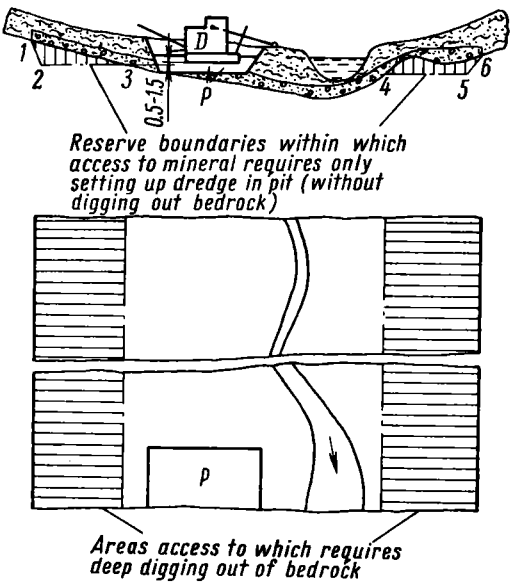


Fig. 329. Opening-up a placer deposit through a pit with and without digging out of the bedrock:
D—dredge; *P*—pit; *1*, *2*, *3*, *4*, *5*, *6*—sections the access to which requires deep digging out of bedrock

In practice, opening-up by a pit is done when the metal is uniformly distributed in drift beds with a very thin peat stratum, in working placers whose underwater depth exceeds 4 m and whose above-water thickness does not exceed 3 m, and in mining predominantly the flood plain portion of the valley.

In opening-up with deep digging (Fig. 329) the dredge also operates in a pit, but has access to areas with high bedrock. The dredge undermines the disintegrated hard bedrock so as to ensure a safe clearance between the pontoon bottom and that of the cut. The digging depth may reach 1.8 m. In this system the bulk output of the dredge is substantially reduced owing to the small struck capacity of the buckets, the cost of repairs increases, and so does the electric power consumption per unit of mined bulk. This is aggravated by increased dilution if the bedrock contains little or no metal.

In practice the deep-digging method of opening-up is applied in the following cases:

when the disintegration of the bedrock extends to a considerable depth and the bedrock carries gruss and contains metal;

in the working of wide-flung placers, when the areas to undergo deep digging are small.

This method is usually employed when digging is done to a depth of up to 1 m.

When opening-up by dams (Fig. 330) the dredge is assembled directly on the ground without excavating a pit. The water basin in which the dredge is to operate is formed by erecting a dam, which

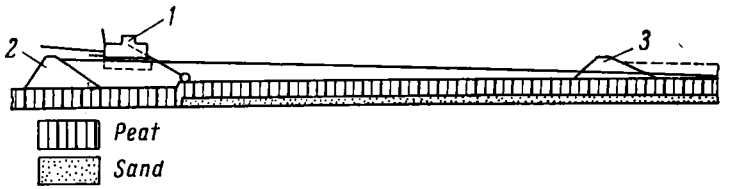


Fig. 330. Opening-up by dams:
1—dredge; *2*—active dam; *3*—new dam under construction

raises the water level in the stream by 2-11 m, usually by 5-7 m. A second dam is constructed at a distance of 0.5 to 10 km which raises the water in the section lying further upstream. The dam consists of an earth embankment and a spillway to let out surplus water at the time of flood. The width of the dam crest is within the range from 3 to 7 m. At the dam site the top soil is removed and an earth fill is constructed by using scrapers and power shovels, or it is built up hydraulically. The dam should be located in the narrower portion of the valley.

This method provides favourable conditions for dredging and secures free access of the dredge to areas where the bedrock is high and the placer is of considerable thickness. There have been examples of mining areas where bedrock elevation above the water reached 3-4 m, with above-water thickness being as great as 10-12 m.

The method has the advantage of extended seasonal dredge-operation time, since the placer surface is flooded and does not freeze in winter months, and then dredging is done in clearer water; also, the coefficient of metal recovery increases. Placer mining with the aid of dams augments the annual yield of washed metal by 15-50%.

Dams should be constructed in as short a time as possible with the use of mechanical equipment and with the corresponding advance over the extraction of sands.

In addition to the above-described methods of opening-up use is made of combined methods, particular preference being given to opening-up by two independent horizons in which the placer is worked out in two benches with separate early development of each. The upper bench is opened up, with an advance, by bulldozers, scrapers or power shovels, while the lower bench, which

includes the metal-bearing part of the placer, is worked by a dredge. The method increases the utilization of the dredge as a mining and concentration unit and cuts down capital outlays per unit of production, since the upper portion of the placer is worked by using less costly, but efficient equipment.

Dredging systems. These are classified according to two features, viz. the number of working faces and the direction of their advance.

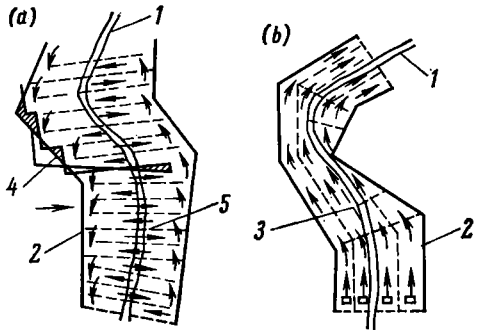


Fig. 331. Dredging methods:
a—with single transverse faces; *b*—with contiguous lengthwise faces; *1*—stream; *2*—boundary of dredge cut; *3*—boundaries of dredge working faces; *4*—lost pillars; *5*—boundary of dredge trips

Two systems prevail in field practice: (1) with single transverse working faces; (2) with contiguous lengthwise working faces (Fig. 331).

The first system. The dredge operates continuously at one face, which advances across the placer. This pattern of mining simplifies the dredge travel, but then, on reaching the placer boundaries, the dredge has to turn through an angle of 180°, and this takes from 4 to 10 hours. The system is commonly employed in working sufficiently wide placers (250-300 m). Its advantages are: the possibility of mining pay areas which can be discovered during exploitation beyond the placer boundaries; high recovery of metal from placers, especially when preliminary exploration was insufficient. In working placers over 7-8 m in thickness refuse dumps usually pile up over the side of the cut, therefore intermediate pillars have to be left between individual dredge cuts to preclude contamination of the placer sands with tailings. These losses may reach 5% depending upon the depth of the placer.

The second system. The dredge operates at several contiguous faces, usually two or three, in some cases even up to six. A placer 100-300 m wide is worked up the rise. In mining by contiguous lengthwise faces no time is lost in turning the dredge; its transfer to contiguous faces takes no more than 15-30 min. At each working face the dredge steps ahead 3-4 times, i.e. it moves over a distance of 10-15 m, and thereafter shifts to an adjacent face. No mineral is lost in intermediate pillars, for the refuse dumps are located at a considerable distance from the working face.

The shortcomings of the method are: the difficulty in working pay areas discovered outside the placer boundaries; the need for a large number of bow lines for the lateral movement of the dredge, and higher expenditures on ice removal in winter time.

Other systems. Apart from the two systems described above some others are also used, for instance the single lengthwise system, which is applied when working narrow placers.

Extraction Methods in Dredging

The following basic methods of extraction (Fig. 332) are used: slicing at a straight working face; slicing at an oblique face; underdigging.

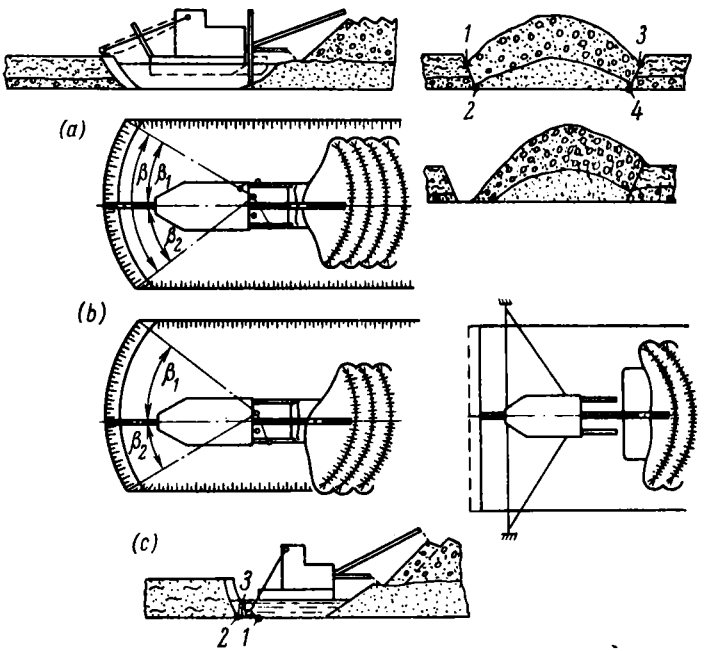


Fig. 332. Extraction methods in dredging:
a—with a straight face; *b*—with an oblique face; *c*—by underdigging

The first of these methods is preferred. The dredge operates in the middle of the dredge cut and to dig the ground it swings right and left through an angle of 35-50°, this ensuring more uniform filling of the buckets. In the course of operation the dredge steps ahead over a distance of 2-7 m, the digging ladder being lifted up to the surface before each dredging trip, and the ground is exca-

vated in slices starting from the surface down to the bedrock. The extracted slice is 25-50 cm thick. With this mode of excavation refuse dumps are sited in the centre of the cut. To avoid piling up the waste over the placer banks, when deposits are of great and medium depth, intermediate pillars have to be abandoned (along lines *1-2*, *3-4*), and this results in losses of up to 5%. In working deep placers there is also a risk of the dredge pontoon contacting the waste dump.

When slicing at an oblique face the dredge operates nearer to that bank of the cut which can be covered with refuse, and it swings through different angles from its central position, thus causing non-uniform filling of the buckets with consequent reduction in dredging-ladder efficiency.

The merits of this mode of extraction are: no refuse piles up over the neighbouring production bank of the placer (no intermediate pillars are left); less danger of the dredge pontoon contacting the waste dump.

In underdigging, the bucket ladder is continuously held on the placer bedrock and the dredge steps ahead over a short distance—within 0.5-1.5 m. The ground is taken out at the bottom of the face along the bedrock, undermining the bench top portion, which caves in and is grabbed by the bucket chain. With this mode of extraction the buckets are filled non-uniformly, the average height of rock lift increases and there is unavoidable mixing of grounds from the upper and lower bench portions, this ruling out selective extraction. This method may be recommended in cases when the mixing of grounds facilitates the process of concentration.

Technical and Economic Characteristics of Dredging

Operating hours per day during the working season—18-20.

Working days per year—160-250 in Eastern Siberia and 250-340 in the Urals. In southern regions the number of working days per year may be increased. In the USSR dredges usually do not operate all year round, the winter months being used for their overhauling.

The output of a dredge varies from 60 thousand to 2.5 million cu m per year, depending upon the bucket capacity and the mining conditions.

The average labour productivity per man engaged in operating a large-capacity dredge is 30-140 cu m per shift. In small-capacity dredges with buckets holding up to 50 litres of the material the output per man drops to 10 cu m per shift.

The average electric power consumption is 1.5-3.0 kW-h per 1 cu m of sand. The total power rating of motors in big dredges reaches 1,200-1,300 h.p.

CHAPTER XIII

HYDRAULIC MINING OF PLACERS

General

By hydraulicking (also called ground sluicing) is meant rending the ground (sands, peats) from its solid mass in place by a strong water jet with subsequent transport of the dislodged material by a stream of water to the site of its treatment or storage.

In hydraulic placering washing of sands in wet jiggers and removal of tailings thereafter are also effected by means of a strong water flow. In hydraulicking water is supplied along pipes to a hydraulic giant (monitor). The water escapes from the nozzle of the monitor at a velocity of up to 20-50 m/sec. In the absence of a valley slope needed to carry away the sands hydraulic elevators or pump dredges are used for lifting the sands to the required height.

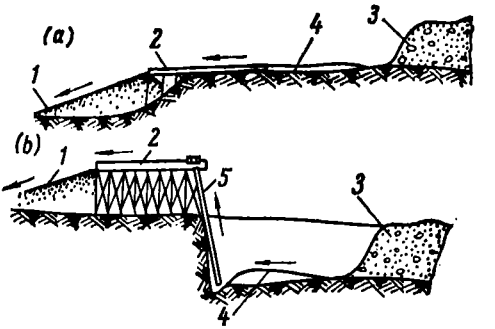


Fig. 333. General patterns of hydraulicking:
a—with pulp carried away by gravity; *b*—by lifting the pulp; *1*—waste heap; *2*—sluices; *3*—placer; *4*—inclined ditch at placer bottom; *5*—hydraulic elevator or dredge pump; arrow indicates direction of sand displacement

The general patterns of hydraulic placering are shown in Fig. 333.

The basic conditions for the application of hydraulic mining of placer deposits are: abundance of water, which is consumed in amounts of 8-60 cu m per 1 cu m of mined ground; the possibility of building up a natural or artificial water head of 2-18 atm; the possibility of unhindered removal of the washed-out product—tailings.

Hydraulic mining was first used in Russia for working gold placers in the Urals as far back as 1830.

Today this method, along with dredging, is one of the principal ways of working placer deposits. Besides placering, hydraulicking is widely employed in the USSR for removal of overburden in surface mining and in construction jobs. In the years to come hydraulicking will undoubtedly gain still more popularity in mining and construction.

Water Supply

In hydraulicking, water supply is a problem of primary importance. Three methods of water supply are distinguished, viz. through ditches, by pumping, and the combined method. Water supply by pumping is practised where electric power is cheap. The pumping plant is ordinarily installed in the valley of the river where the placer is located, the water being delivered to the production face along pipes. In areas lacking cheap electric power water has to be supplied through ditches over a distance of tens of kilometres. In the absence of a river, or with insufficient volume of water therein, it must be collected by means of dams, which are projected on the basis of studies into the meteorological and hydrographic conditions existing in the area concerned. In most cases earth-fill dams built up of sandy-clayey ground are used. Small dams are made of fimber logs and bunched fagots. An opening is provided in dams for an outlet pipe with a gate to regulate the discharge of water and another pipe to drain off excess water.

Water from storage reservoirs, or from the point where it comes from the river, is delivered to the consumer along ditches, wooden flumes, troughs or pipelines.

Ditches. These are the cheapest to construct, and therefore they are used whenever the local conditions are favourable. Ditches are made of trapezoidal shape with their side-wall slopes corresponding to the angle of natural repose for the ground. The bottom width of ditches should be 1.8-2.2 times their depth. In soft ground the side walls of the ditch usually have a slope of 1 : 1, in strong ground 1/2 : 1 and in rocky ground 1/4 : 1. The longitudinal gradient of ditches depends on the permissible velocity of the flow, which is somewhat lower than that at which the ground is subject to washout.

Permissible water flow velocity in a ditch, m/sec

Silt	0.15
Fine loam	0.20
Sandy ground	0.40
Loose ground with coarse gravel	0.75
Heavy ground, loam	0.90
Gravel	1.08

Heavy ground with coarse gravel . . .	1.60
Crushed stone	1.70
Soft basement rock	2.00
Conglomerate, soft shale	1.8-2.1
Laminated ground	2.1-2.7
Hard rock	3.6-4.5

Earth ditches are commonly excavated with a gradient of 0.0005 to 0.0025, in individual cases up to 0.004.

Water losses in ditches (through seepage and evaporation) amount to 10-30%. Seepage losses depend upon the total area of the wet

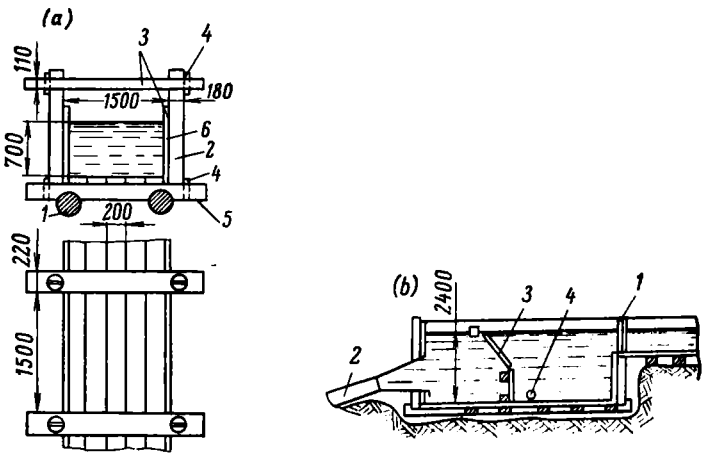


Fig. 334. Water supply arrangement:
a—wooden flume: *1*—girders; *2*—posts; *3*—carrier; *4*—wedges; *5*—sill; *6*—boards (lagging); *b*—pressure tank: *1*—gate; *2*—pressure pipe; *3*—protective screen; *4*—outlet opening

surface of the ditch and the transmission coefficient of the ground. The highest seepage losses occur in sandy ground, reaching 0.5 cu m per day from 1 sq m of wet surface of the ditch.

Flumes. This is a wooden trough intended for the passage of water; it is set up on timber posts, piles, cribs or laid directly on the ground surface (Fig. 334*a*).

Flumes are commonly made of boards 50 mm thick. The rate of water flow in flumes is 1.5-2.5 m/sec, which is higher than in ditches. Flumes have a grade of 0.0015-0.002, with the width of the trough being twice the height of the water stream. The principal dimensions of the trough and its grade are calculated by using the same equations as in the case of earth ditches. The setting up and maintenance of flumes are more expensive than for ditches and

therefore they are employed when water has to be carried over a valley, ravines or on a steep side-hill, or when water seeps intensively through wall rocks, or when the latter tend to wash out rapidly.

Pressure tank (Fig. 334*b*). This is a wooden tank set up at the end of the ditch and provided with a protective screen to retain objects clogging the delivery pipe and also with outlet openings for discharging mud. Since the ditch gradient (0.0005-0.0025) is several times less than that of the river (0.005-0.015) from which water is drawn and along whose ridge the ditch runs, there is a considerable difference between the elevation marks of the pressure tank and the placer being mined. The greater this difference, the higher the hydraulic head.

Pipes are used to deliver water from the pressure tank to consumers devices. Pipes used in hydraulicking may have a diameter of 200-1,000 mm (metal) and 800-1,000 mm (wooden). Wooden pipes are employed when the water head does not exceed 6 atm. The rate of water flow in pipes varies from 1.5 to 5 m/sec.

The combined method of water supply is used extremely rarely.

Hydraulic Giant Construction and Operation

Water from the pressure tank or a centrifugal pump is fed to the hydraulic giant. The purpose of the giant is to point a dense water jet with an outflow velocity of 20-50 m/sec at the working face. The giant (Fig. 335) has the following basic parts: the lower bend fixed to a foundation beam or skids, the upper bend with a joint, the bell-mouth, the barrel with a nozzle and the control level with a counterweight.

In powerful units and with high heads a special device—deflector—is used to change the direction of the jet; the deflector is located between the barrel and the nozzle, it rotates freely on a hinge and is controlled by a separate lever. The latter is handled with ease and ensures accurate control of the hydraulic giant. In recent years giants with remote control have been manufactured in the USSR.

Soviet-made hydraulic giants have an outlet opening of 100-300 mm designed for nozzles 25-125 mm in diameter. Usually 50-100-mm nozzles are used. The effective range of the hydraulic giant jet depends upon the water head. Optimal distances between the monitor and the working face in relation to the mining conditions have been established empirically. When the unit operates solely to disintegrate and dislodge the material the distance between it and the working face should not exceed one-quarter of the water head value in metres; if the purpose is dislodgement and concurrent

transport of the material this distance should be equal to one-third of the water head, and when the aim is only to transport the ground the distance may be equivalent to the range of the jet throw.

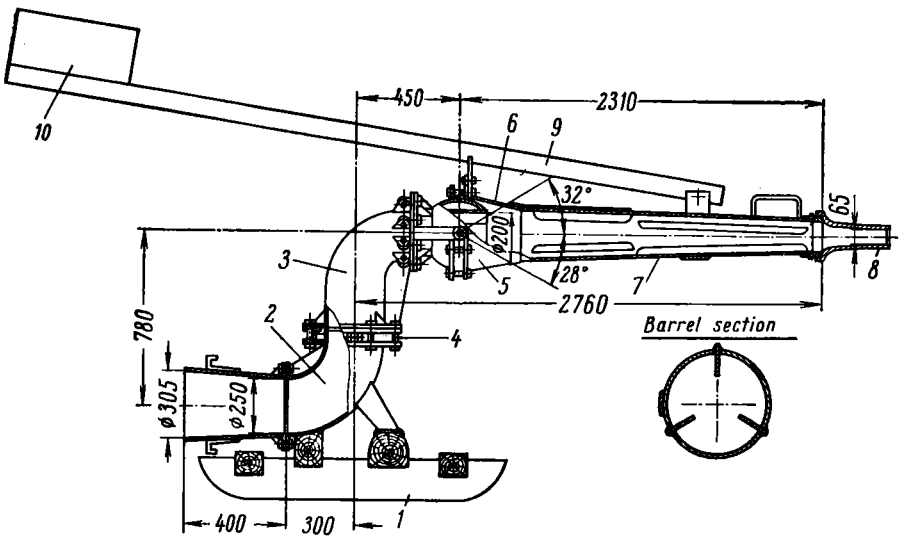


Fig. 335. ГМН-250 hydraulic giant (welded, total weight 300 kg):
1—skids; *2*—lower bend; *3*—upper bend; *4*—king bolt; *5*—joint; *6*—bell-mouth; *7*—giant's barrel; *8*—nozzle; *9*—control lever; *10*—counterweight

The velocity of water issuing from the giant's nozzle is determined by the equation

$$v = \varphi\sqrt{2gH}$$

where v = rate of water outflow, m/sec;
H = head at nozzle, m;
g = acceleration (9.81 m/sec^2);
φ = speed factor (it depends upon the nozzle design, φ = 0.94-0.97).

The required diameter of the nozzle is estimated according to the equation

$$d = \sqrt{\frac{1.27Q}{\mu v}}$$

where d = diameter of nozzle outlet opening, mm;
Q = water consumption, m^3/sec;
v = rate of water outflow, m/sec;
μ = coefficient of jet compression (μ = 0.96-0.98).

With a definite nozzle diameter and velocity of outflowing water its consumption can be estimated from the equation

$$Q = \mu \frac{\pi d^2}{4} v, \ \text{m}^3/\text{sec}$$

The giant's efficiency largely depends upon the properties of the ground washed, the height of the working face, the water head and the nozzle diameter.

Table 44 lists approximate giant's efficiency standards, allowance being made for the dislodgement and transport of ground to the washery (with a water head of 35 m and a working face 4-5 m high).

Table 44

Hydraulic Giant's Efficiency Standards in Cubic Metres of Ground in Place During 1 Hour of Continuous Operation

Category of ground	Nozzle diameter, mm							
	50		63		75		100	
	Water consumption, litres/sec	Efficiency, m³/hr	Water consumption, litres/sec	Efficiency, m³/hr	Water consumption, litres/sec	Efficiency m³/hr	Water consumption, litres/sec	Efficiency, m³/hr
I	49	14.6	78	23.4	112	33.8	196	59.0
II	49	8.8	78	14.0	112	20.5	196	35.4
III	49	4.9	78	7.8	112	11.2	196	19.6
IV	49	2.71	78	4.32	112	6.2	196	10.9
V	49	2.52	78	4.0	112	5.76	196	10.0

Category I includes peat with no roots, loose top soil, loose sandy-clayey ground;
category II—sandy pebbles or clay-cemented tough ground containing some pebble and coarse gravel (up to 30%);
category III—tough clays with boulders up to 50 cm in diameter amounting to 15% of the total, clay-bonded debris of bedrock, broken shales;
category IV—tough clays with boulders over 50 cm in diameter amounting to 3% of the total, unbroken marl and clay-cemented sandstones; weakly cemented conglomerates; frozen ground up to 30%;
category V—very tough clays with 50% of boulders over 50 cm in diameter, semi-broken sandstones; frozen ground up to 50%.

Operational Set-up in Hydraulicking

Opening-up. In ground sluicing two basic methods of opening-up are employed, viz. through a washout ditch and through pits.

Opening-up through a washout ditch (Fig. 336) is practised in the working of bench placers with a plot of low land in the vicinity

suitable for gravity disposal of tailings. In this method a washout ditch is excavated on the side of the ridge so that the ditch can approach, at a design grade, the most distant placer to be mined through the ditch. The ditch gradient should vary from 0.017 to 0.035, depending upon the size and roundness of the gravel being transported and the height of the water stream section (water discharge in m^3/sec). In opening-up through a washout ditch expenses on transporting the sands during mining are insignificant and

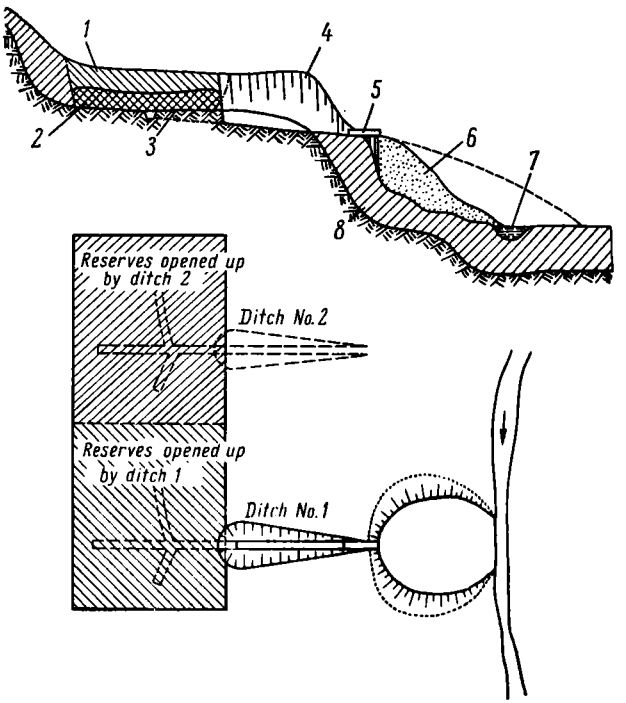


Fig. 336. Opening-up through a pilot washout ditch:
1—peats; *2*—sands; *3*—bedrock; *4*—ditch; *5*—sluice; *6*—waste dump; *7*—river; *8*—silting deposits

the cost of their extraction is low. The cost of driving washout ditches depends upon the gradient of the underlying bedrock in the placer. It begins to rise when the bedrock grade is less than 0.02 and in this case the washout ditch must be excavated deeper in the bedrock (even down to 7 m if the mining conditions are difficult). On a higher level and at a distance of 100 to 1,000 m from the first washout ditch, depending upon the local conditions, yet another ditch is driven, which serves to work out a higher-lying

area. The size of the areas to be mined influences the depth to which the ditch must penetrate into the bedrock and the capacity of dump sites for tailings, which should be located on the flood plain.

Opening-up through a pit (Fig. 337). This mode of opening-up involves excavation of a pit, with transport equipment placed therein; a sluice for washing of sands is made near it. In the course of placering, as the working face moves further away from the pit,

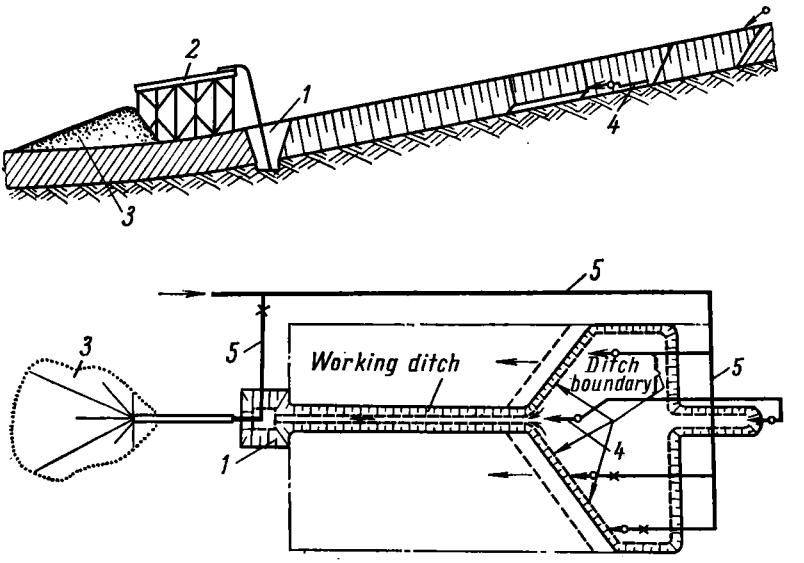


Fig. 337. Opening-up through a pit with concomitant mining:
1—pit; *2*—sluice; *3*—waste heap; *4*—hydraulic giant; *5*—water pipeline

the dredge pump is shifted each time over a distance of 50-100 m, and only then is a mud-settling sump dug out, near which the dredge pump is set up. When the latter is moved far away from the pit and the whole dump-site capacity for the disposal of tailing has been used up, the washery, together with the delivery pipeline for sands, is shifted nearer to the working face. The washery is moved over a distance of up to 70-250 m. With this mode of opening-up consumption of electric power to bring the sand up reaches 3-8 kW-h per 1 cu m of ground, the dredge pump is subjected to considerable wear and tear, and the net cost of winning sands is higher than with the first method of opening-up. This method is employed when the placer occurs in particularly difficult mining conditions, with no ground nearby suitable to accommodate facilities for gravity placering and when the bedrock grade is small. The method usually

finds application in the working of placers occurring in the flood plain of valleys and on low-lying terraces with irregular bedrock consisting of limestones. Among the combined methods the method with separate opening-up of two horizons is the most popular, and in this case the placer is worked by two benches, with the upper bench mined through a washout ditch, and the lower through a pit.

Systems of hydraulic mining. These are classified according to the way in which the giant jet is used in moving washed-out ground from the working face to the main transport facilities. In hydraulicking of placers preference is given to the methods with concomitant mining, with a side face and a fan-shaped face.

In the first system (Fig. 337), after the pit has been excavated a working ditch is driven along the rise of the placer over a distance of 40-120 m. Once the water pipeline has been laid and the ditch completed its upper portion is widened up to the placer boundary and the hydraulic giants are moved onto the bedrock. This is followed by all-round washout of the ground in the working face along the dip of the placer. With this mode of stopping the water jet flows in the direction of sand displacement, this considerably reducing consumption of water per unit of sand transported from the working face to the dredge pump, and the efficiency of the giant and dredge pumps rises. Among the disadvantages of the system is the necessity of driving a working ditch; the amont of labour consumed in driving it increases with the depth of the placer. The system is used in working placers not more than 7-8 m deep and 40-50 m wide.

The lateral-face method (Fig. 338). A bank of the pit is washed along the rise of the placer to make it wide enough to reach the placer boundary, following which the equipment is moved onto the bedrock. One or two advance headings are first driven in the face, usually along the banks of the open-cut, at sites where the bedrock occupies the highest position. The advance headings are driven over a distance of 6-12 m. When excavating the advance heading the direction of the washing jet is opposite to that of the sand flow and this appreciably increases unit consumption of water. Later on the monitor is moved to the upper part of the advance heading and the ground in the central portion of the face is worked out by the method involving concomitant actual mining with reduced unit consumption of water for sand washing. This system does not require working ditches. On the whole the method with a lateral working face involves higher water consumption than the method with concomitant actual mining. It is suitable for the working of placers not more than 7 m deep, when the use of the method with concomitant advance of working faces presents difficulties.

The fan-shaped-face method (Fig. 339). In this method the working ditch is driven across the placer. Hydraulic giants set up in the

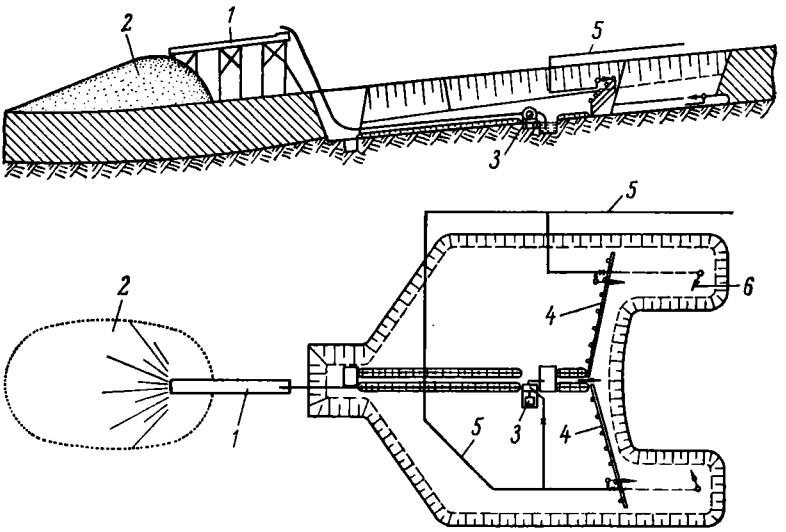


Fig. 338. Lateral-face method:
1—sluice; *2*—waste heap; *3*—dredge pump; *4*—shield; *5*—water pipeline; *6*—hydraulic giant

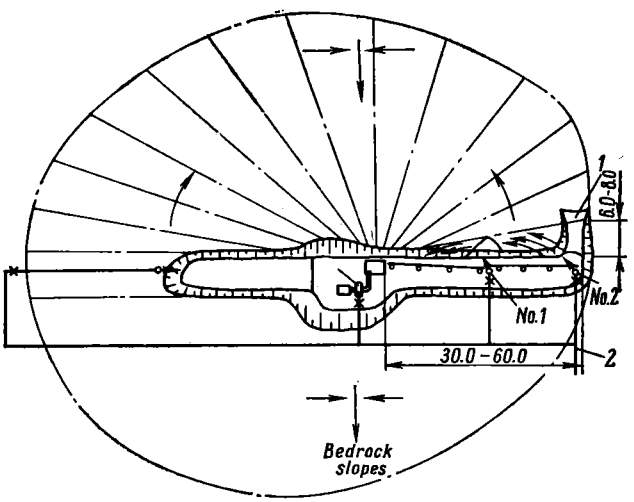


Fig. 339. Fan-shaped-face method:
1—leading cut; *2*—water pipeline; *3*—dredge pump; Nos. 1 and 2—hydraulic giants

ditch wash the ground at the face non-uniformly, tending to accelerate washout in the section of the face farthest from the pit.

The face progresses in a fan-like fashion, the mud-settling sump in the pit forming the centre of the fan. The angle of the face slew varies from 70 to 360° (usually it is 160-220°). As regards unit water consumption and the conditions for its application, the system is similar to the lateral-face method.

Methods of sand wash. The following methods of sand wash are used: with a hydraulic giant set up on the bedrock and with bench

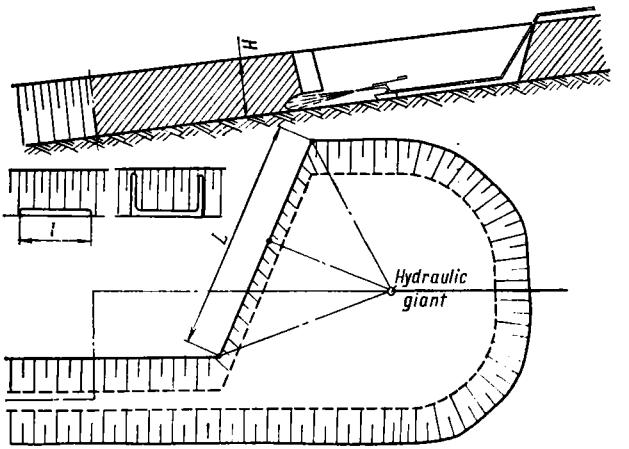


Fig. 340. Washout with bench undercutting

undercutting; with mechanical loosening of the ground; with abandonment of unwashed ground.

The first method (Fig. 340) finds the widest application. Here the monitor rests on the bedrock. The bench is first undercut over a distance of 15-30 m so as to cause mass caving of the face top. Then the ground is moved towards the mud-settling sump of the dredge pump. The method is practised in grounds of hardness not above the 3d category and with benches not exceeding 3 m in height. For safety sake the hydraulic giant should be put up at a distance from the face breast not smaller than the height of the bench. After the ground has been washed off over a distance of 2 to 8 m the hydraulic giant is moved on to the working face, and the ground washing process is repeated.

In washout with mechanical loosening of the ground (Fig. 341) the latter is first ripped by a bulldozer or a power shovel. This method is employed when the hardness of the ground exceeds 1.5. Preliminary ripping of the ground reduces the water consumption in

bench undercutting. With benches up to 5 m high bulldozers are commonly used to rip the ground. They pile up the material near the bench toe where the giant is installed. When the bench is high this is done by an excavator, with the pile of loosened ground placed on top of the bench (in the case of a dragline) or near its toe (in the case of a power shovel). This method is more efficient with stronger rocks and expensive electric power. Water consumption is reduced by 30-50%.

In the third method (Fig. 342) the bench is undercut with abandonment of a sand layer on the bedrock so as

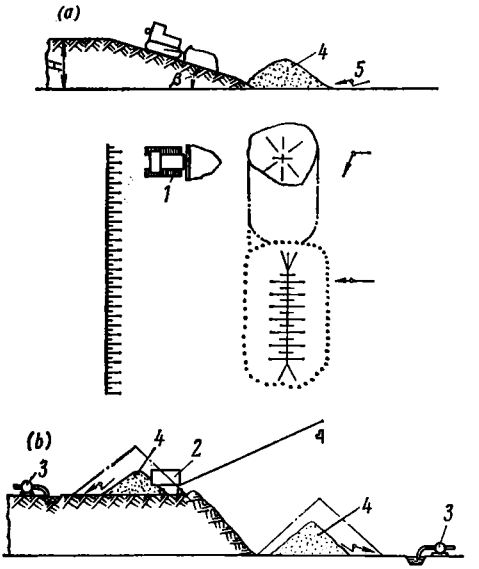


Fig. 341. Washout with mechanical loosening of ground:
a—with the use of bulldozer; *b*—with the use of power shovel; *1*—bulldozer; *2*—power shovel; *3*—dredge pump; *4*—ground to be washed out; *5*—hydraulic giant

to increase the gradient of the ground wash-off surface. The height of the unwashed ground should be such that the wash-off surface slope is close to 0.02. An increase in the gradient considerably reduces the amount of water consumed for the transport of sands over the major part of the face. The remaining layer of unwashed ground, 0.5-2 m thick, is carried away by the giant jet, aided by a bulldozer. Ground washout with part of it left unwashed finds application in working placers with very slightly sloping bedrock (less than 0.005), and also when the bedrock has crevices which can be penetrated by metal.

Fig. 342. Washout with abandonment of unwashed ground:
1—unwashed ground; *2*—bulldozer; *3*—dredge pump

Transport of Washed Sands

The best way to transport sands from the working face to the washery is by gravity, provided the surrounding area has suitable gradients. To wash away fine-grained material, a slope of 0.02 will

suffice, while for medium-sized material it should be 0.04-0.05.

If the required gradients are not available the washed sands should be transported to the washery with the aid of hydraulic elevators and dredge pumps. The former is a water-jet pump adapted for lifting sand mixed with water.

The hydraulic elevator (Fig. 343) operates essentially as follows: water at a head is brought along the delivery pipe to a nozzle mounted in the hydraulic elevator. Like in the hydraulic giant, a jet of water escapes at a high velocity, which depends upon the head and flows to a receiving mixer chamber. Having passed through the throat the jet enters a diffuser and then the pulp pipeline. Entraining the air from the receiving chamber the water jet sets up a vacuum, and owing to this the pulp entrained by the jet begins

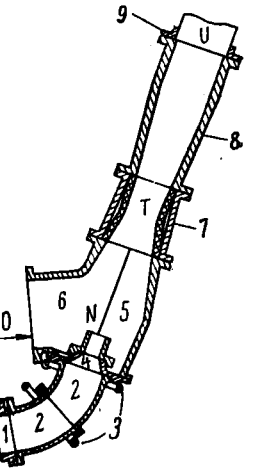


Fig. 343. Hydraulic elevator:
1—delivery pipe flange; *2*—swivel elbows; *3*—flanges; *4*—nozzle butt; *5*—ground section; *6*—entrance section; *7*—intake opening; *8*—outer section; *9*—upcast pipe flange; *O*—intake opening; *N*—nozzle; *T*—throat; *U*—upcast pipe

to fill the suction pipe, and after passing through the throat and diffuser is transported along the pulp pipeline. Thus, in the hydraulic elevator transportation of pulp is realized by the force of the water jet, on which the amount of the pulp lifted and the height of the lift depend.

To prevent ingress into the elevator of large stones which could cause its stoppage, a retaining screen is set up at the elevator mud-settling sump. The elevator is commonly tilted at an angle of 35-45°. With correctly designed individual elements of the elevator the pulp lift height varies between 1/4 and 1/6 of the water head at the nozzle; (usually it is 6-7.5 m, occasionally 10-12 m).

A serious disadvantage of the hydraulic elevator is its low efficiency. The weight of the solid material lifted by the elevator constitutes not more than 5% of its total lifting capacity. Hydraulic elevators are employed when there is abundance of water at a natural head sufficient to lift the pulp to a height of 7-8 m.

The dredge pump has a higher efficiency than the hydraulic elevator. It is a centrifugal pump designed for the pumping of water mixed with ground. The dredge pump rotor has two or three blades with clearances between them wide enough to let through chunks

of ground. The body of the dredge pump is protected on the inside by a lining of interchangeable steel sheets.

Below we give the characteristics of a Soviet-made dredge pump:

Output (in terms of water), litres/sec	100-400
Head, m	20-26
Motor rating, kW	44-200
Branch pipe diameter, mm	
suction	200-350
delivery	150-300
Maximum size of rock pieces admitted, mm	100-200

In the USSR still heavier dredge pumps are manufactured.

Technical and Economic Characteristics of Hydraulic Plants

The output of individual enterprises using the hydraulic method of mining equals 70-2,500 thous. cu m per year. Greater tonnages are handled by hydraulic plants with water supplied through ditches, and smaller tonnages, with water fed by pumps.

The minimum service life of a hydraulic plant varies between 3 and 5 years, depending upon the existing conditions, the normal service life is 10-15 years.

The average water consumption is 20-40 cu m per 1 cu m of ground. At a low water head, in rubbly placers with a small slope of the underlying bedrock and with the use of water for the lifting of sands, water consumption rises to 100 cu m per 1 cu m of ground.

The desirable bedrock gradient in the direction of the washaway is 0.02-0.4. With smaller slopes water consumption rises and so do the expenses on washing off the ground.

Output per man engaged in hydraulicking is 17-30 cu m per shift, depending upon the scope of operations; the average output is 25 cu m per manshift.

At high-capacity hydraulic plants and under favourable conditions of hydraulicking the output per manshift may reach 40-50 cu m and even more.

Electric power consumption in hydraulically operated open-pits with artificially built-up water head constitutes 5-15 kW-h per 1 cu m of mined bulk.

General Conclusions on Hydraulic Mining

Under the appropriate conditions hydraulicking is one of the most efficient methods of mining.

Ground sluicing, particularly with preliminary breaking and loosening of ground, may be employed with a great economic effect

in winning other kinds of useful minerals, as well as gold, both with the opencast and the underground methods of mining. Hydraulic removal of overburden is successfully used in ore open-pits at a number of Soviet mines.

Experimental hydraulicking in underground conditions yielded good results in a number of mining industries (coal, gold placers, manganese ore, etc.).

The Scraper-Bulldozer Method of Placer Mining

It this method peat and sand are taken out by bulldozers and wheel scrapers. The usual bucket capacity of scrapers is 6-10 cu m.

Scraper-bulldozer units are chiefly employed in the working of permafrost placer deposits occurring to a depth of not more than 12-14 m. In Magadan and Yakutia up to 50% of the entire metal is won by this method of mining. Wide-scale application of scraper-bulldozer units in the working of frozen sands is due to the fact that here extraction of sands is done by slices as they thaw out and with no expenditures on breaking the ground. Bulldozers serve to move the ground over a distance of not more than 100-170 m with the total digging depth of up to 2-2.5 m. The pressure exerted by the machine on the ground is about 0.5 kg/cm^2. Scrapers move the ground over a distance of 150-600 m with the total digging depth of up to 6-8 m. The pressure exerted by the machine on the ground is 2 kg/cm^2.

Dewatering. To raise the efficiency of machines, ensure better work conditions and reduce losses during the mining of a placer, it must be previously drained, with water diverted outside the open-cut.

Dewatering is effected in the same way as in hydraulic mining, i.e. by means of stream-course diversion, uphill and permanent ditches, the latter being deepened down to the placer bedrock.

Opening-up with Scraper-Bulldozer Units

Opening-up by two separate horizons is the most popular method (Fig. 344). The upper horizon is opened up by continuous ramp roads or by individual outgoing ramp roads spaced at 50-100 m. The ramp road gradient varies from 0.15 to 0.25. Figure 344 shows a continuous ramp road. It is used when the thickness of the active bench does not exceed 2.5 m. With a greater thickness separate outgoing ramp roads are made; these are generally used with scraper stripping. The lower horizon is opened up through a pit in which a sand-receiving bin and a small pumping plant are set up. Sands are transported to the washery on the surface on a conveyer, which is installed in a small trench connecting the pit with the ground sur-

face. In bulldozer mining pits are spaced at about 200 m, while in extraction by scrapers the spacing is increased to 400-500 m. The pit

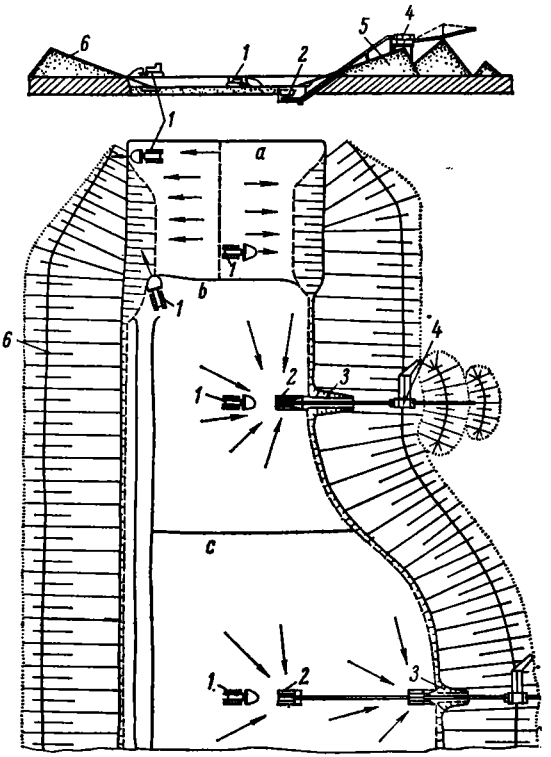


Fig. 344. Opening-up by two separate horizons and the fan-shaped-face method of mining sands:

a—opening-up of the top horizon by continuous outgoing road and stripping in parallel cuts; *b*, *c*—opening-up of bottom horizon through pit, and fan-shaped-face method of mining sands; *1*—bulldozer; *2*—pit; *3*—trench; *4*—washery; *5*—washed sand dump; *6*—peat dump

is excavated by drilling and blasting with the use of a rope-slusher hoist.

The systems and methods of ground extraction in scraper and bulldozer work differ from one another, therefore they will be considered separately.

The Scraper Methods of Mining

In this method two basic systems are used, viz. with one-way and two-way travel (Fig. 345). In the first system the scraper delivers one bucket from the open-cut to the dumping site in a round

trip. The material can be brought to the ground surface along one or two open-cut banks. This system is used predominantly, it is applied in working placers more than 50 m wide. With two-way travel the scraper delivers two buckets to the dumping site in one round trip. This reduces the number of turns the scraper must make and the time spent on moving one bucket.

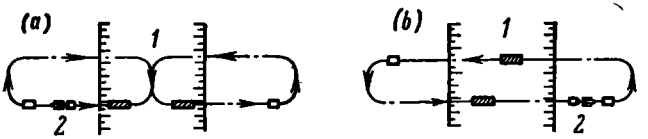


Fig. 345. Methods of scraper mining:
a—with one-way travel; *b*—with two-way travel; *1*—cut; *2*—scraper

The two-way travel system is chiefly practised in the working of narrow placer deposits, especially when stripping is done by travelling along continuous ramp roads.

The scraper travel paths may take the shape of a loop or of the figure eight.

Methods of Scraper Digging

Three ways of digging are the most popular: by crest-shaped strips, with preliminary ripping, and with the use of a pusher (Fig. 346).

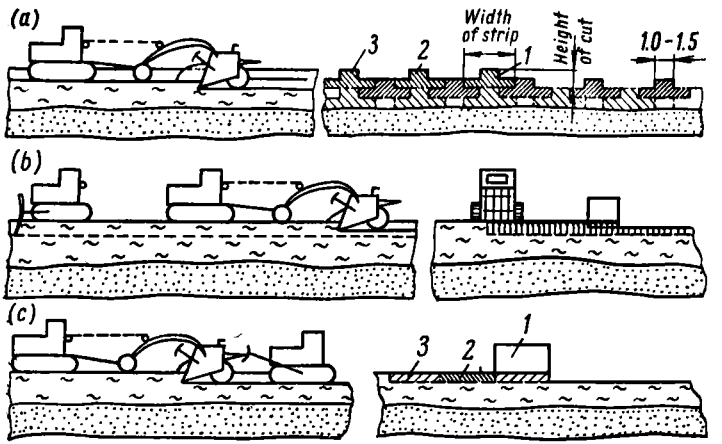


Fig. 346. Methods of scraper stripping:
a—with crest-shaped strips; *b*—with preliminary ripping; *c*—with the use of a pusher

In digging by crest-shaped strips the scraper operates so that a pillar 1-1.5 m wide is left between the individual strips. The under-

cut layer is crest-shaped. This method allows the effective charge ratio of the bucket to be increased in working grounds with a coefficient of hardness $f = 0.7\text{-}1.0$.

Extraction with preliminary ripping is practised in rocks with a coefficient of hardness $f = 1.5\text{-}2.0$. In this case a special tractor-

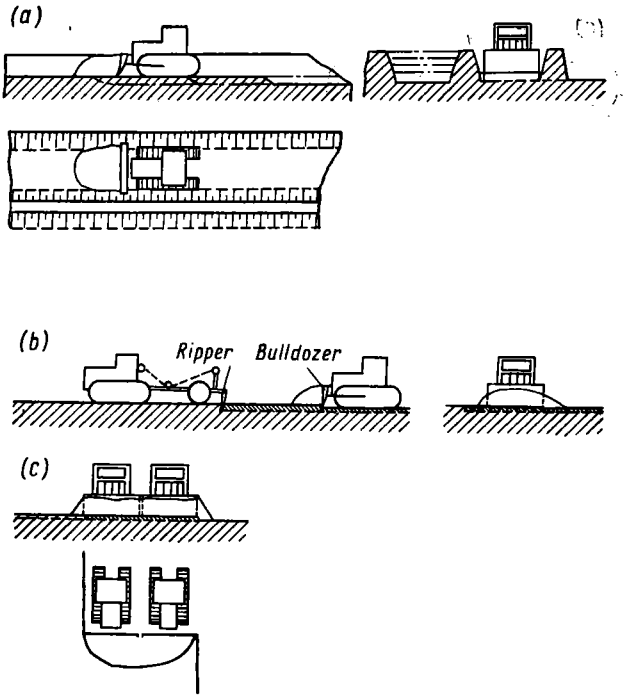


Fig. 347. Methods of stripping by bulldozers:
a—furrow (trench) method; *b*—with preliminary ripping; *c*—by twinned machines operating in tandem

ripper breaks the ground preliminarily to a depth of up to 40 cm, following which the scraper grabs and moves the loosened material.

In stripping with a pusher, when the scraper bucket is being filled, a special tractor-pusher, sometimes a bulldozer, is driven to the tail end of the scraper and pushes the scraper, thus helping to undercut a thicker layer of ground and increase the effective charge ratio of its bucket. This method is used in the working of grounds with a coefficient of hardness $f = 1\text{-}1.5$. One pusher can take care of 3-5 scraper units. The ground can be stripped either in horizontal or in inclined strips.

Methods of Mining by Bulldozers

When winning sands the bulldozer moves them to a bin along the shortest route, and therefore the pattern of bulldozer travel routes is fan-shaped with the bin in the centre (Fig. 344). In removing overburden the parallel system of stripping is mainly used with travel routes running parallel to one another.

Methods of Stripping by Bulldozers

Three methods of stripping are used most extensively, viz. by trenches, with ripping of the ground, and by employing twinned machines (Fig. 347).

In the first method the bulldozer does the stripping by moving along one and the same stretch of ground, thus digging a furrow 0.5-1.5 m deep. This method enables the largest bulk of ground to be moved with minimum losses, which raises the efficiency of the bulldozer. The hauls usually exceed 40-50 m. The pillar left between the furrows is recovered at a later date. This method of stripping is practised in the working of grounds with a coefficient of hardness $f = 0.7\text{-}1.5$.

The second method is employed in the working of heavier grounds ($f = 1.5\text{-}2.0$) and they must first be loosened by a ripper. Stripping and moving of the ground is done after it has been loosened.

In the third method 2-4 bulldozers are placed in a single line with a 10-15-cm clearance left between their mouldboards. The machines move at the same speed, this helping to bunch a considerable bulk of material, thus raising the efficiency of the scraper by 25-30%. The method is also employed in stripping overwetted ground with haul distances of over 100 m.

Basic Technical and Economic Characteristics of Scraper-Bulldozer Methods of Mining

Working days per year in sand winning—100-130.

Working days per year in stripping—120-160.

Annual output of bulldozer—up to 40 thous. cu m, and that of scraper—up to 45 thous. cu m.

Output per man engaged in scraper-bulldozer work: in stripping—60-90 thous. cu m per shift; in winning of sands—30-35 thous. cu m per shift.

CHAPTER XIV

NEW METHODS AND BASIC OBJECTIVES IN THE WORKING OF MINERAL DEPOSITS

New Methods for the Working of Deposits

In assessing the efficiency of the methods for the working of mineral deposits currently applied one must emphasize their general shortcomings, namely the complexity of work, high labour requirements, much time needed to prepare the deposit for actual mining, considerable initial outlays, etc., all this being particularly true of underground methods of mining.

Today research workers and engineers are searching for novel, improved methods of mining, which differ radically from those practised until now.

One of such methods—underground leaching of copper from the ore with subsequent recovery of copper from solutions on the ground surface—is discussed below.

Underground gasification is not considered in this book, since it is rarely used.

Leaching of Copper and Other Metals from the Ore

The winning of useful minerals by dissolution and subsequent recovery has been practised on an industrial scale for a good many years. Examples are the winning of rock salt by its dissolution and subsequent evaporation, the winning of sulphur by its dissolution in superheated water and subsequent isolation by cooling. In many cases water is used to obtain suspended pulp and then precipitate kaolin therefrom. Recovery of copper from solutions (acid mine water) was originally done in the XVI century, but its wide-scale application began only in the XX century.

At present the USSR has a large number of installations for the recovery of copper from mine water containing this metal in the form of $CuSO_4$ in amounts of up to 3-5 (on the average 1-2) g/litre.

A simple and cheap process of copper extraction from mine water by cementation on the ground surface proceeds according to the reaction

$$CuSO_4 + Fe = FeSO_4 + Cu$$

Copper is precipitated in troughs filled with iron scrap, along which copper-bearing mine water flows. Copper can also be precipitated in vats filled with iron scrap, where water is changed periodically. Copper is discharged from the troughs and vats at regular intervals. Depending upon the quality of the scrap, the composition of the mine water and the proper conduct of the process the cement copper contains an average of 70% of pure copper and up to 30% of iron and other impurities. Not less than 90% of the copper is extracted from solutions and, under favourable conditions, this figure may be raised to 98-99%. An average of 1.2 kg of scrap is consumed to recover 1 kg of copper.

The simplicity and economic efficiency of obtaining copper from mine water served as a basis for introducing a special method for the working of copper deposits by underground leaching with subsequent recovery of copper from solutions on the ground surface. Copper is leached from the deposit by the action of sulphuric acid, which is formed as a result of the ingress of water, specially supplied from the ground surface, into some sections of the deposit. The copper solution is passed along ditches in mine workings to special collectors, wherefrom it is delivered by pumps to the ground surface, where the copper is precipitated from the solution.

The following conditions are necessary for working copper deposits by this method: solubility of copper in a weak solution of sulphuric acid; permeability of the solid ore for the solvent (the presence of joints and fissures, crushed ore structure); the absence of channels which would permit the solvent to flow without sweeping through the whole mass of the solid ore in situ. Experience shows that the latter two conditions can be provided artificially.

Ore minerals of other useful components (lead, zink, nickel, etc.) are insufficiently soluble in a weak solution of sulphuric acid, and the possibility of utilizing special solvents for them is now under study. The following can be used for recovery of copper by leaching: deposits with lean copper ores whose exploitation by ordinary methods is unprofitable; danger areas with undrawn crushed ore; fire-hazard areas; areas with crushed pillars; old mines with incompletely worked-out mineral reserves; dump-piles of copper-containing waste on the ground surface.

In the USSR there are many deposits which can be worked by leaching.

Underground leaching requires driving a certain number of mine-openings to irrigate producing sections and take off the water. The number of these openings can be reduced to a minimum by drilling irrigation boreholes at definite intervals.

In Soviet mines recovery of copper from the ore by dissolution with subsequent precipitation is mostly done concurrently with

underground mining by ordinary methods with the use of mine waters flowing through the deposit.

It should be noted that the technology of underground leaching is in its initial stage of development. The prospects for obtaining metals, above all of copper, directly from the deposit are, however, very bright. The results obtained so far prove the expediency of wider introduction of underground leaching of copper ores and the continuation of research on ores of other metals.

In the last few years great achievements have been attained in the field of underground leaching of uranium ores by using two methods:

(1) forcing a 3% solution of sulphuric acid through boreholes from the ground surface and pumping out the uranium-enriched solution to the ground surface via special wells (leaching of uranium-bearing sandstones);

(2) irrigation with a 3% solution of sulphuric acid of solid uranium ore in place broken up with the aid of explosives, with subsequent diversion of the uranium-enriched solution along special underground workings to the shaft, through which it is pumped up to the ground surface and then directed to a hydrometallurgical plant.

The prospects for the use of nuclear charges for the fragmentation of lean ores will greatly increase the commercial importance of underground leaching, especially in connection with the possibility of bacterial leaching of ores at a small cost.

Major Objectives in the Working of Mineral Deposits

The working of mineral deposits is a complex process. The most important problems to be solved in this connection are: the choice of the mining method (opencast or underground), the adoption of the most rational variant of early development; the selection of an effective mining system.

In handling these problems many factors should be taken into consideration, the most important of them being the geological, mining and economic conditions of working the deposit. Lack of comprehensive geological data rules out the correct solution of the basic problems and impedes the preparation of a satisfactory project for mining the deposit. It should be borne in mind that geological data submitted for the preparation of a project serve as a basis for adopting definite methods of mining, the pattern of early development and the mining system; it is sometimes entirely impossible to introduce changes into them, or such changes may entail considerable additional expenses and loss of time. The wrong solution of basic problems directly affects the economics of the

mining enterprise and jeopardizes the safety of operations; it increases initial outlays, delays commissioning of the mine and raises the cost of mining because of the disagreement between the envisaged and actual conditions. Comprehensive study of the mining and geological conditions is the prime duty of the geological and prospecting staff.

Among the immediate problems the proper solution of which is important for the further technical and economic development of many mining enterprises are:

a considerable increase in labour productivity at all stages of mining and a reduction in the mining cost through the use of improved methods of work and mining systems, all-round mechanization of operations, automatic control of equipment, extensive use of continuous conveyer transport, mobile machinery, and efficient organization of work based on the achievements of progressive workers; in underground mining particular attention should be given to the wide utilization of systems involving bulk breakage of ore by longholing and the use of mobile equipment;

a detailed study and mastering of rock pressure control, scientific substantiation of parameters of mining systems and of the sequence to be adopted in extracting blocks, improvement and introduction (under suitable conditions) of systems involving the use of compact (hydraulic and self-setting) mine-fill;

elimination of the gap between the rate of stoping and development work by broad introduction of high-speed drifting methods; the overall intensity of the working of mineral deposits should be sharply increased in the near future;

reduction in the losses and dilution of minerals and observance of the stipulated quality standards as to the grades and classes of the mineral; properly organized sorting of the mineral mined and strict fulfilment of the requirements of processing technology, including selective extraction of different grades of the mineral when necessary;

elimination of fire hazards in coal and chalcopyrite deposits;

devising expedient methods for mining at great depth under increased rock pressure and with sudden outrushes of coal and gas;

introduction of new methods of mining, such as manless extraction, underground leaching of metals from the ore, underground hydraulicking, the use of mobile equipment, etc.;

large-scale introduction of effective methods of open work involving the use of up-to-date drilling, loading, and transport equipment; opencasting by course stacking with inside waste-dumps and ground sluicing, conditions permitting; in opencast mining particular attention should be given to the correct determination of the

marginal stripping ratio and the optimal depth of the open-pit;

large-scale application of dredging and hydraulicking in the working of placer deposits.

Problems of Safety in the Working of Mineral Deposits

The basic information on the safe working of mineral deposits has been given in the relevant sections of this book. Below is a brief summary on safety in mining.

A. Measures to ensure safety in underground mining of deposits:

proper delineation of rock-shifting and caving zones;

adoption of mining systems and selection of their design features in accordance with the mining and geological characteristics of the deposit;

strict supervision of the observance of permissible exposure areas in the back and walls of the mined-out space with the use of appropriate methods of rock-pressure control;

timely drainage of producing sections of the deposit and precautions against inrushes of quicksand and other types of waterlogged ground;

provision of production stopes with two exits to the main horizons and to the ground surface;

observance of the established sequence of extracting blocks, pillars, rooms, stumps, sublevels and individual slices in the production level;

observance of the established rules for the conduct and organization of blasting operations;

the use of durable mine support structures with wedging and lagging of the roof and walls in workings;

implementation of the necessary fire-prevention measures, especially with systems involving extensive use of mine timber, and in working chalcopyrite and coal deposits; these include the use of fire-proof timbering, extension of water supply lines, stone development, observance of the rules for the use of electric power, etc;

effective ventilation and cooling of air at deep-lying horizons;

dust control in mine workings, provision of sufficient supply of air and proper lighting and adoption of special protective measures in the presence of methane and in the working of sections hazardous in respect to sudden outrushes of coal and gas.

The control of spontaneous outrushes of coal and gas and the elimination of coal dust are problems of prime importance which should be solved in the immediate future.

B. Measures to ensure safety in opencast mining:

routine control over the stability of the rock in individual benches and banks of the open-pit;

checkup calculations to verify the stability of the earth masses (see courses of engineering geology) and rigid control of the observance of the prescribed sizes of benches, protective berms and areas and slope angles;

timely dewatering of the open-pit and protecting it against flooding with storm, spring and subsurface waters;

observance of the stipulated track gradients and train traffic rules;

observance of the rules for the protection of workers against injuries by electric current during the operation of electric excavators and locomotives;

conduct of blasting operations with due regard for safety rules concerning the testing, storage and use of explosives—delineation of danger zones and organization of blasting;

effective ventilation of deep open-pits.

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